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Haile Gold Mine
Lancaster County, South Carolina

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OceanaGold Corporation

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Appendices

Appendix A: Certificates of Qualified Persons
Appendix B: LoM Annual Cash Flow Forecast

Appendix C: Taxation Model

1 Summary

This National Instrumented 43-101 (NI 43-101) Technical Report (Technical Report) was prepared for OceanaGold Corporation (OceanaGold) to a feasibility level by SRK Consulting (U.S.), Inc. (SRK) on the Haile Gold Mine (Haile or Project). This report includes both open pit and underground mining components and a single economic analysis based on open pit and underground reserves.

1.1 Property Description and Ownership

Haile is 100% owned and operated by OceanaGold. Haile is located 5 kilometers (km) northeast of Kershaw in southern Lancaster County, South Carolina. The Haile property site is 30 km southeast of Lancaster, the county seat, and 80 km northeast of Columbia, the state capital of South Carolina. Geologically, Haile is situated in the Carolina terrane, which also hosts the past-producing Ridgeway, Brewer and Barite Hill gold Mines. The Carolina terrane was the location of the first gold rush in the United States in the early 1800s.

1.2 Geology and Mineralization

Haile is the largest gold deposit (4.36 million ounce (Moz) resource) in the eastern USA. Haile consists of eleven named en echelon gold deposits within a 3.5 km by 1 km area. Haile occurs within a strongly deformed ENE-trending structural zone of the Carolina terrane at or near the contact between metamorphosed volcanic and sedimentary rocks of Neoproterozoic to Early Cambrian age. Mineralization is exposed along an ENE-trending anticlinorium. Deformation is dominantly ductile with ENE fold axes and isoclinal folds overprinted by shearing and local brecciation. Haile is hosted in foliated siltstones and greywackes of the upper Persimmon Fork Formation. Recent reinterpretation of stratigraphy at Haile considerably simplifies the structural model with a folded volcanic-sedimentary package that is not complicated by overturning or regional thrusting. The conformable ENE-trending contact between the Persimmon Fork and the overlying Richtex Formation is located about 0.5 km south of Haile.

1.3 Status of Exploration; Development and Operations

Resource definition drilling at Haile by Romarco Minerals Inc. (acquired by OceanaGold in October 2015) and now OceanaGold has increased the reserves almost five-fold since 2007. Reserve growth resulted from 3D geologic modelling, higher gold prices and deeper drilling of a robust and previously underexplored mineral system. This has been recently exemplified by pre-development of the Horseshoe underground deposit and announcement of a maiden reserve in this report. Aggressive drilling by OceanaGold continues at a rate of ~one km of drilling per week targeting open pit and high-grade underground targets mineralization proximal to the sedimentary-volcanic contact. Underground development of the Horseshoe deposit in 2019 to 2020 will facilitate access for underground drill stations along the prospective one km long Horseshoe-Palomino trend. Additionally, regional exploration target generation and land acquisition for Haile-like deposits has accelerated with the goal of discovering >0.5 Moz Au within 30 km of Haile.

1.4 Mineral Processing and Metallurgical Testing

Samples of ore were collected by Haile Gold Mine, Inc. (HGM) for metallurgical testing which indicated that the ore will respond to flotation and direct agitated cyanide leaching technology to extract gold.

Comminution test work on mineralized samples was performed by RDi, and ALS Limited (ALS). Tests included Bond work indices and Sag Mill Comminution (SMC) and JK Drop Weight impact testing. The results of the test work were used to develop the expanded plant comminution circuit design.

Laboratory testing on ore composite samples demonstrated that the mineralization was readily amenable to flotation and cyanide leaching process treatment. A conventional flotation and cyanide leaching flow sheet can be used as the basis of process design.

The relative low variability of test work indicates that the different mineralized zones are similar in terms of ore grindability, mineral composition, and flotation and cyanide leaching response.

Overall gold recovery will be in the range of 65% to 92% dependent primarily on head grade to the mill and less related to which zone the ore is mined from.

The data developed in the test programs has been used to establish a relationship between overall gold recovery and head grade.

1.5 Mineral Resource Estimate

1.5.1 Open Pit Mineral Resource Estimate

The Mineral Resources at Haile are comprised of both potential open pit and underground ores. Separate block models were generated for the open pit and then underground areas. As such, the open pit and underground Mineral Resources are described separately.

At this time, there are no unique situations relative to environmental or socio-economic conditions that would put the Haile Mineral Resource at a higher level of risk than any other developing resource within the United States.

The open pit Mineral Resources are shown in Table 1-1.

Table 1-1: Open Pit Mineral Resources as of January 1, 2017

Resource Summary - HGM16 model - US$1500/oz shell - 0.45 g/t			
HGM16	**Tonnes (Mt)**	**Au (g/t)**	**Au (Moz)**
Measured	7.06	1.97	0.45
Indicated	52.2	1.63	2.73
Inferred	11.1	1.4	0.49
Measured and Indicated	59.2	1.67	3.17

Source: OceanaGold

- CoG = 0.45 g/t Au and US$1500/oz.
- Tonnages are metric tonnes.
- Grades are in g/t
- Open pit Mineral Resource is reported within a US$1,500/oz optimized shell.
- Mineral Resources in this table includes the Mineral Reserve.
- The open pit Mineral Resources were estimated by Bruce van Brunt, a Qualified Person.

1.5.2 Underground Mineral Resource Estimate

The resource estimation is based on the current drill hole database, interpreted lithologies, geologic controls and current topographic data. The resource estimation is supported by drilling and sampling current to November 1, 2016. The estimation of Mineral Resources was completed utilizing a computerized resource block model constructed using Vulcan™ modeling software.

The current geologic model is not yet conclusive on the age of gold mineralization with respect to the regional folding. Silicified ore zones would have acted as rigid bodies during deformation and hence preferentially occur in tightly folded rocks in anticlines. Preferred meta-siltstone host rocks are related to more brittle geology during early deformation compared to the overlying volcanic units. Healed fault and breccia zones and quartz vein swarms commonly host gold in the upper zone, and are interpreted to reflect post-gold structural foci that utilized pre-gold structural zones of weakness.

In order to isolate mineralized material from non-mineralized, SRK constructed ARANZ Leapfrog Geo software (Leapfrog) implicitly modeled wireframe solids which enclose anomalous gold mineralization above a 1.25 grams per tonne (g/t) Au threshold. The grade shells were constructed using interpreted trend planes of mineralization. The trend planes were developed by digitizing section profiles of gold continuity which were then triangulated into 3D planes of gold continuity. There are two dominant zones of mineralization; an upper moderately northwest dipping zone and a lower near vertical zone.

For all estimations, the following criteria were used:

- Dynamic search orientation essentially parallels to the plane of gold continuity for each zone;
- Minimum of three composites and maximum of eight composites to estimate grade;
- Sample length weighting to account for any short composites located at the ends of drill holes;
- Composites from a minimum of two drill holes; and
- Composites from a minimum of two octants.

As part of the grade estimation, model validation is conducted as an interactive process. Six techniques were used to evaluate the validity of the block model. These included:

- A visual check of composite versus block grades on sections, plan views and in 3D;
- A review of the general model estimation parameters;
- Statistical comparison between the estimated block grades to the composite sample data
- A nearest neighbor estimation comparison; and
- An assessment of the impacts of edge dilution about the margins of the Au grade shell.

The results of the validations are all in compliance to industry standards and support a robust estimation.

The Mineral Resources reported for the Horseshoe deposit are classified as Indicated and Inferred Mineral Resources, based primarily on drill hole spacing since all other supporting data is of good quality. A wire frame solid was constructed around the areas where the majority of the blocks were estimated in the first or second pass of the estimation which had an average search distance of less than 10 meters (m) and 15 m respectively. These wireframe solids were used to assign the Indicated Mineral Resource classification. All blocks outside of the Indicated wireframes were classified as Inferred Mineral Resources.

The Horseshoe Mineral Resource statement is based on the Ordinary Kriging (OK) model as presented in Table 1-2. The resource is not confined within any conceptual stope designs and a CoG of 1.17 g/t Au has been applied. The CoG assumes underground mining methods and is based on a mining cost of US$35/t, milling cost of US$10/t, administration cost of US$5.60/t, a gold price of US$1,500/oz, and a gold recovery of 90%.

Table 1-2: Horseshoe Underground Mineral Resource Statement as of November 1, 2016, SRK Consulting (U.S.), Inc.

Classification	Au Cut-Off (g/t)	Tonnes (Mt)	Au (g/t)	Contained Au (Moz)
Indicated	1.17	2.7	5.68	0.50
M & I	1.17	2.7	5.68	0.50
Inferred	1.17	1.2	5.0	0.20

Source: SRK, 2016

- Mineral Resources are not ore reserves and do not have demonstrated economic viability.
- All figures rounded to reflect the relative accuracy of the estimates.
- Metal assays were capped where appropriate.
- CoG is based on a mining cost of US$35/t, milling cost of US$10/t, administration cost of US$5.60/t, a gold price of US$1,500/oz. and gold recovery of 90%.
- The underground Mineral Resources were estimated by Bart Stryhas, a Qualified Person.

1.5.3 Combined Open Pit and Underground Resource Estimate

Table 1-3 presents the combined open pit and underground resource statement for the Haile Property.

Table 1-3: Combined Open Pit and Underground Resource statement

Mining Type	Resource Category	Au Cut-Off (g/t)	Tonnes (Mt)	Au (g/t)	Au Contained (Moz)
Open Pit	Measured	0.45	7.06	1.97	0.45
Underground	Measured	-	-	-	-
Subtotal	*Measured*	-	*7.06*	*1.97*	*0.45*
Open Pit	Indicated	0.45	52.2	1.63	2.73
Underground	Indicated	1.17	2.71	5.68	0.49
Subtotal	*Indicated*	-	*54.9*	*1.83*	*3.22*
Subtotal	**M & I**		**61.9**	**1.84**	**3.67**
Open Pit	Inferred	0.45	11.1	1.4	0.49
Underground	Inferred	1.17	1.2	5.0	0.20
Subtotal	**Inferred**		12.3	1.7	0.69

Source: SRK, 2016

- The open pit resource is as of January 1, 2017. The underground resource is as of November 1, 2016.
- The open pit Mineral Resources were estimated by Bruce van Brunt, a Qualified Person. The underground Mineral Resources were estimated by Bart Stryhas, a Qualified Person.
- Mineral Resources are not ore reserves and do not have demonstrated economic viability.
- All figures rounded to reflect the relative accuracy of the estimates.
- Underground CoG is based on a mining cost of US$35/t, milling cost of US$10/t, administration cost of US$5.60/t, a gold price of US$1,500/oz. and gold recovery of 90%.
- Open pit CoG is 0.45 g/t Au and US$1,500/oz, reported within a US$1,500 optimized shell.
- Mineral Resource includes the Mineral Reserve.

1.6 Mineral Reserve Estimate

1.6.1 Open Pit Mineral Reserves Estimate

The sub-blocked geological model was regularized to a standard block size. The dilution that was introduced by this process resulted in a 1% increase in ore tonnes and a 2% reduction in gold grade. No further ore losses or ore dilution were applied.

The open pit Ore Reserves are reported within a pit design based on open pit optimization results. The optimization included Measured, Indicated and Inferred Mineral Resource categories with a gold price of US$1,300/oz Au. The reserve pit used to define reserves was based on a US$1,150/oz Au pit shell as the basis of mine design. Subsequent to pit optimization, Inferred material (approximately 13%) within the reserve pit was treated as waste and given a zero gold grade. The overall pit slopes used for the design are based on operational level geotechnical studies and range from 45° to 70°.

Measured Mineral Resources were converted to Proven Mineral Reserves and Indicated Mineral Resources were converted to Probable Mineral Reserves by applying the appropriate modifying factors, as described herein, to potential mining pit shapes created during the mine design process.

The open pit mine design process results in open pit mining reserves of 55.0 Mt with an average grade of 1.71 g/t. The Mineral Reserve statement, as of January 1, 2017, for the Haile Open Pit is presented in Table 1-4.

Table 1-4: **Haile Open Pit Mineral Reserves Estimate as of January 1, 2017**

Description	Tonnes (Mt)	Au (g/t)	Au Contained (Moz)
Proven	7.55	1.97	0.48
Probable	47.50	1.66	2.54
P+P	55.00	1.71	3.02

Source: SRK

- Reserves are based on a US$1,300/oz Au gold price.
- Open pit reserves are stated using a 0.45 g/t cut-off and assume full mine recovery.
- Open pit reserves are not diluted (Further to dilution inherent to the resource model and assume selective mining unit of 5 m x 5 m x 5 m).
- Metallurgical recoveries are based on a recovery curve (1-(0.2152*au grade^-0.3696))) that equate to an overall recovery of 82%.
- Reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- Mineral Reserves were estimated by Bret C Swanson, BE (Min) MMSAQP #04418QP, a Qualified Person.

SRK knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors that might materially affect the underground Mineral Reserve estimate.

1.6.2 Underground Mineral Reserves Estimate

The Project is currently being mined as an open pit mine. The mineralization extends down at depth and outside of the pit extents. This mineralization, not mined by the open pit, is assessed as an underground mine and is referred to as Horseshoe.

Based on the orientation, depth, and geotechnical characteristics of the mineralization, a transverse sublevel open stoping method (longhole) with ramp access is used. The stopes will be 15 m wide and stope length will vary based on mineralization grade and geotechnical considerations. A spacing of 25 m between levels is used. Cemented rock fill (CRF) will be used to backfill 75% of the stopes and non-cemented waste rock will be used in the remaining stopes. The CRF will have sufficient strength to allow for mining adjacent to backfilled stopes.

For the feasibility study (FS) design, the deposit has been divided into two major production areas. The “upper area”, which extends from the -70L to the top of the orebody, will be mined from bottom to top, with stoping initially commencing on the -70L. A permanent sill pillar has been designed between the -70L and -75L. The "lower area" will also be mined from bottom to top, with stoping commencing on the -240L and progressing upward until the permanent sill pillar on the -75L is reached.

A 3D mine design has been created representing the reserve areas. The underground mine design process resulted in underground mining reserves of 3.1 Mt (diluted) with an average grade of 4.38 g/t.

This estimate is based on a mine design cut-off of 1.5 g/t. The numbers include a 93% to 100% mining recovery based on type of opening (Stope, development, etc.) to the designed wireframes in addition to a 0% to 7% unplanned dilution. A portion of the dilution does include grade as described in section 15.2.2. An additional development allowance of 16% was applied to main ramps and 5% to level accesses to account for detail currently not in the design.

Mineral reserves were classified using the 2014 CIM Definition standards. The Mineral Reserve statement, as of January 1, 2017, for the Haile Horseshoe Underground is presented in Table 1-5.

Table 1-5: **Haile Horseshoe Underground Reserves Estimate as of November 1, 2016**

Description	Tonnes (Mt)	Au (g/t)	Au Contained (Moz)
Proven	-	-	-
Probable	3.12	4.38	0.44
P+P	3.12	4.38	0.44

Source: SRK

- Reserves are based on a gold price of US$ 1,300/oz. Metallurgical recoveries are based on a recovery curve (1-(0.2152*au grade^-0.3696)) that equates to an overall recovery of 88%.
- Underground reserves are stated using a 1.5 g/t cut-off. Mining recovery ranges from 93% to 100% depending on activity type. Mining dilution is applied, partially at zero grade and partially using a grade calculated from a representative section. The dilution ranges from 0% to 7% depending on activity type.
- Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- Mineral Reserves have been stated on the basis of a mine design, mine plan, and cash-flow model.
- The Mineral Reserves were estimated by Joanna Poeck, BEng Mining, SME-RM, MMSAQP #01387QP, a Qualified Person.

1.6.3 Combined Open Pit and Underground Reserves Estimate

Table 1-6 presents the combined Open pit and underground reserves statement for the Haile Property.

Table 1-6: Reserve Statement for OceanaGold's Haile Gold Mine

Mining Type	Resource Category	Tonnes (Mt)	Au (g/t)	Au Contained (Moz)
Open Pit (OP)	Proven	7.55	1.97	0.48
Underground (UG)	Proven	-	-	-
Subtotal	*Proven*	*7.55*	*1.97*	*0.48*
Open Pit	Probable	47.5	1.66	2.54
Underground	Probable	3.12	4.38	0.44
Subtotal	*Probable*	*50.6*	*1.83*	*2.98*
Total (OP + UG)	**P+P**	**58.2**	**1.85**	**3.46**

- The open pit reserve is as of January 1, 2017. The underground reserve is as of November 1, 2016.
- Mineral Reserves are based on a gold price of US$1,300/oz. Metallurgical recoveries are based on a recovery curve (1-(0.2152*au grade^-0.3696)) that equates to an overall recovery of 82% for the open pit material and 88% for the underground material.
- Open pit Mineral Reserves are stated using a 0.45 g/t Au cut-off and assume full mine recovery. Open pit reserves are not diluted (Further to dilution inherent to the resource model and assume selective mining unit of 5 m x 5 m x 5 m)
- Open pit reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- Underground Mineral Reserves are stated using a 1.5 g/t Au cut-off. Mining recovery ranges from 93% to 100% depending on activity type. Mining dilution is applied, partially at zero grade and partially using a grade calculated from a representative section. The dilution ranges from 0% to 7% depending on activity type.
- Mineral Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- Mineral Reserves have been stated on the basis of a mine design, mine plan, and cash-flow model.
- The open pit and underground Mineral Reserves are effective as of January 1, 2017. The open pit Mineral Reserves were estimated by Bret C Swanson, BE (Min) MMSAQP #04418QP, a Qualified Person. The underground Mineral Reserves were estimated by Joanna Poeck, BEng Mining, SME-RM, MMSAQP #01387QP, a Qualified Person.

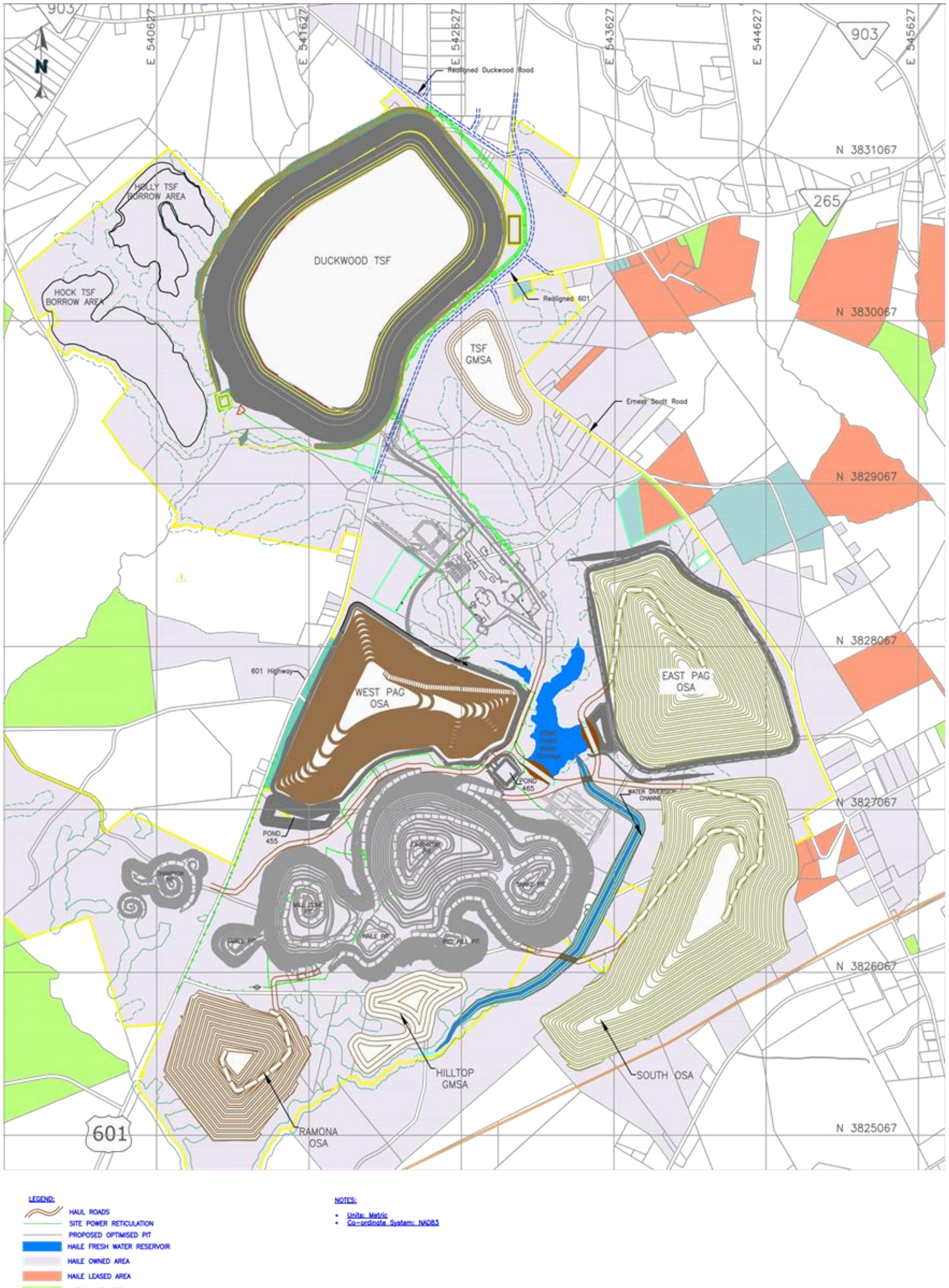
1.7 Mining Methods

1.7.1 Open Pit Mining Methods

Haile is currently being mined using conventional open pit methods. The open pit is located in gently undulating topography intersecting natural drainages that will require diversion to withstand high rainfall events during the summer months. Waste dumps are managed according to the AP and are located on high ground and lined for control of drainage (contact water) if required.

The open pit that forms the basis of open pit reserves and life of mine (LoM) production schedule is approximately 2.5 km from east to west, 1.25 km north to south with a maximum depth of 370 m. Total material movement is estimated at 536 Mt comprised of 55 Mt of ore and 481 Mt waste giving a strip ratio of 8.74 (Waste:Ore). Ore grade averages 1.71 g/t Au yielding over 3 Moz of gold in situ. SRK have created 13 pit/pushbacks that are approximately 150 m to 300 m wide with the minimum mining width of 60 m. Bench sinking rates are approximated to one 5 m bench per month.

Figure 1-1 illustrates the site layout and final pit design.

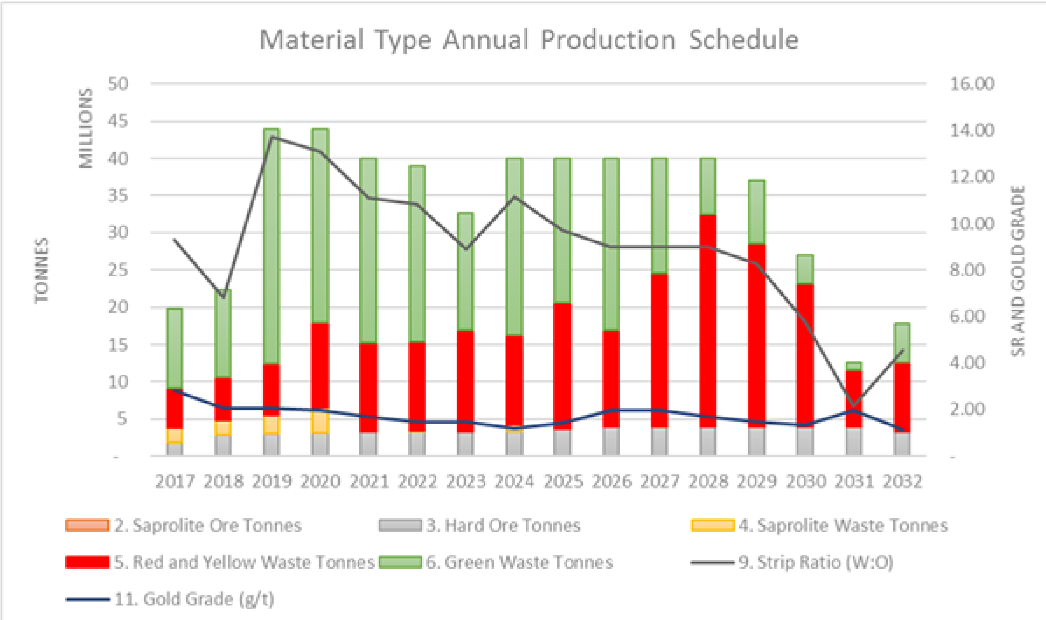


Source OceanaGold

Figure 1-1: Final Pit Design and Site Layout

2017 production rates are 2.1 million tonnes per annum (Mtpa) of ore delivered to the mill and 22.3 Mtpa of total material movement. Over the next 18 months, open pit production will ramp up to 3.3 Mtpa of ore and 44.0 Mtpa of total material movement. Open pit ore production will increase to 4.0 Mtpa in 2027, after the Horseshoe underground has been exhausted.

The mine production schedule is summarized in Figure 1-2.



Source: SRK

Figure 1-2: LoM Production Schedule

The Haile open pit equipment currently being used on site has been kept for the ore mining (Caterpillar (Cat) 6020B excavator and Cat 777 haul trucks) on 5 m benches. As total production ramps up, larger Cat 6030 excavators and 785 haul trucks will be used to supplement the current fleet. A Cat 993 front-end loader (FEL) or similar sized loader will act as backup to the hydraulic excavators. Table 1-7 lists the major production equipment expected for the next eight years, but a full fleet estimation with sustaining capital and rebuilds has been included in the economic model. A down the hole explosive service will continue to be provided by a contractor.

Table 1-7: Major Equipment for the first eight years

Fleet	2017	2018	2019	2020	2021	2022	2023	2024
Cat 6020	1	1	1	1	1	1	1	1
Cat 6030			2	2	2	2	2	2
Hit 1900 / Cat 993	2	2	2	1	-	-	-	-
Trucks 777 - Ore	12	12	4	4	4	4	4	4
Trucks 785 - Waste	3	3	18	21	13	15	20	20
Ore Drill - Cat 5150C	1	1	1	1	1	1	1	1
Waste Drill - Cat MD6290	2	2	4	4	3	4	3	4

Source: OceanaGold

1.7.2 Underground Mining Methods

Geotechnical

A geotechnical field characterization program has been undertaken to assess the expected rock quality. This program included logging core, laboratory strength testing, in situ stress measurements and oriented core logging of jointing. The results of this program have provided adequate quantity and quality data for feasibility-level design of the underground workings.

A geotechnical assessment of the orebody shape and ground conditions has determined that long-hole open stoping mining is an appropriate mining method. Stopes have been sized to maintain stability once mucked empty. A primary/secondary extraction sequence with tight backfilling allows optimization of ore recovery while maintaining ground stability. Primary stopes will be backfilled with cemented rockfill, while secondary stopes will be backfilled with uncemented waste rock.

The design has been laid out using empirical design methods based on similar case histories. The stability of the design has been checked with 3D numerical stress-strain models of the workings which included consideration for mine-scale faulting. The modeling results confirm that stopes and access drifts are predicted to remain stable during active mining.

Mine Design

Stope optimization within Vulcan software was used to determine potentially economically minable material. The mining method is transverse sublevel open stoping with cemented rockfill (CRF). Stope sizes, used in the optimization were 25 m high, 15 m wide and a minimum of 10 m long.

Stope optimizer shapes were used as a basis for the design work. Each stope has a 5 m x 5 m access located at the bottom of the stope. Top accesses (also 5 m x 5 m) are designed to give access to stopes on the next level and to allow for backfilling. The stopes are drilled from the top and rings are blasted from the end of a stope toward the footwall access. The blasted material is remotely mucked from the stope access. A primary/secondary stoping sequence will be used. The stope accesses are connected to a level access located in waste material. The level accesses connect to the main ramp which is located in the footwall. Each level access is connected to an intake and exhaust ventilation system. Ore will be remotely mucked from the bottom stope access using a 14-t LHD (6.2 m^3) and loaded into 40-t trucks for haulage to surface.

The mine is accessed via decline and two raises (5.0 m diameter, blind sink and/or raisebored) connect the mine to the surface (one intake, one exhaust). The intake raise is outfitted with a hoisting system for emergency egress.

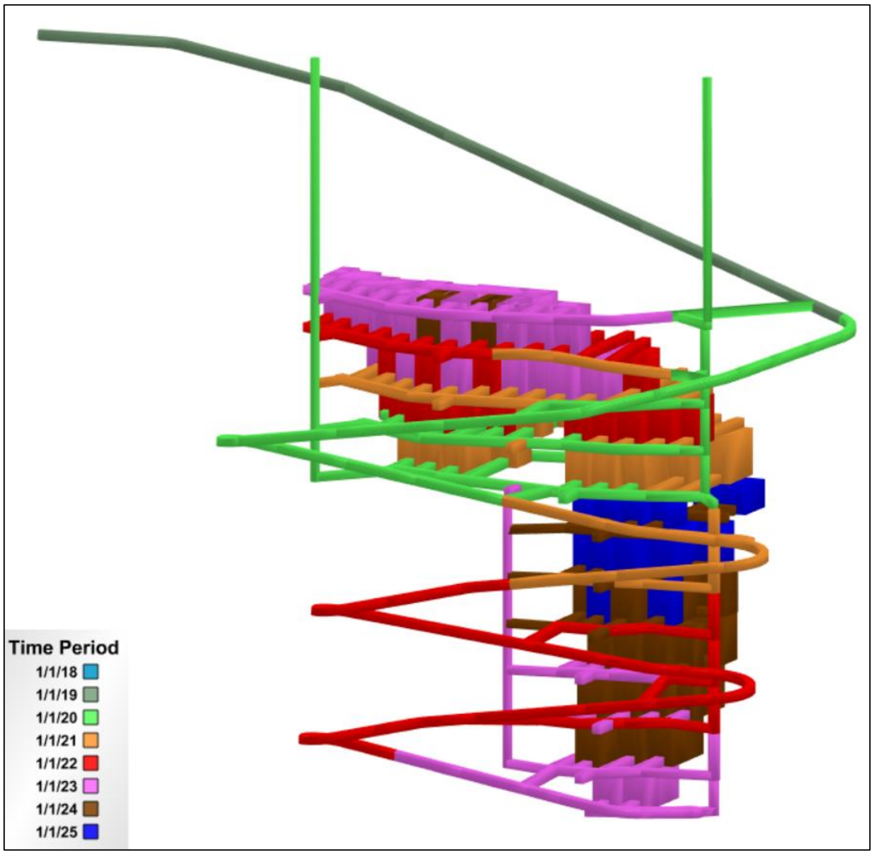
The production and development schedule was completed using iGantt software. A delay of seven days was used prior to operating equipment on CRF and a 14-day delay prior to mining adjacent to a CRF filled stope. A production rate of 1,923 t/d was targeted with ramp-up to full production as quickly as possible.

The current assumption is the successful permitting of underground mining by mid-2019 with portal development beginning July 1, 2019. The open pit is more than able to meet this timeframe for stripping and opening the portal bench area. Portal development is assumed to take six weeks after which mining of the decline can begin. First production from the stopes occurs in December 2020 and lasts through mid-year 2025. Table 1-8 shows the yearly production schedule and Figure 1-3 shows the mine production schedule colored by year.

Table 1-8: Horseshoe Mine Production Annual Mining Schedule

Year	Mineralized Tonnes (kt)	Au (g/t)	Waste Tonnes (kt)	Backfill Volume (m^3)
2018	-	-	-	-
2019	-	-	47.9	-
2020	44.6	4.54	181.4	3,647
2021	701.1	4.42	95.6	223,912
2022	701.1	4.77	116.7	252,960
2023	701.2	4.03	113.8	249,620
2024	703.2	4.46	28.6	248,038
2025	265.0	3.95	0.0	106,909
Total	**3,116.0**	**4.38**	**584.0**	**1,085,086**

Source: SRK



Source: SRK

Figure 1-3: Mine Production Schedule Colored by Year

1.8 Recovery Methods

The processing methods will remain the same as the currently operating plant for the proposed expansion. A conventional flotation and cyanide leaching flow sheet will continue to be used at Haile.

Additional equipment will be installed in some areas of the plant to meet the expanded duty and some reconfiguration of existing apparatus will be completed.

The key additions to the process plant for the expanded capacity required include:

- Secondary crushing and screening;
- Pebble crushing installation on existing Semi-Autogenous Grinding (SAG) mill scats recycle;
- Tertiary (whole ore) grinding using a vertical ball mill;
- New primary regrinding stage using a vertical ball mill;
- New pre-aeration tank with oxygen injection;
- Expanded concentrate leaching; and
- Additional thickening capacity.

As well as these key changes, throughout the plant general pump and piping upgrades are required.

1.9 Project Infrastructure

The permitted Duckwood Tailings Storage Facility (TSF) will be expanded to store plant tailings by raising the crest height. The permitted Johnny's PAG (JPAG) Overburden Storage Area (OSA) will be expanded to store additional PAG (East PAG) material and an additional PAG OSA (West PAG) will be developed to store the remaining PAG material that will be generated by the pits. The water from upper Haile Gold Mine Creek (HGMC) will be diverted around the proposed pits via a diversion channel. The existing 69 kV electrical system will require upgrades to meet the site power demand.

The underground infrastructure required to support underground mining will include general buildings, upgrade and extension of the power lines and water supply. An underground run-of-mine (RoM) pad area will contain the stockpiles, CRF plant as well as a truck shop and laydown area.

1.10 Environmental Studies and Permitting

In late 2014, HGM received all of the major permits required for construction and operation of the current mine plan, including: Clean Water Act 404 Dredge and Fill Permit, Mine Operation Permit, 401 Water Quality Certification, TSF Dam Permit, North Fork Dam Permit, Air Permit (to construct), and various National Pollutant Discharge Elimination System (NPDES) Permits (wastewater treatment and storm water).

Haile is unique in that it occurs wholly on private land owned by HGM and does not impact federal/public (United States Department of the Interior Bureau of Land Management (BLM) or United States Forest Service (USFS)) lands that would be subject to projected modifications from these surface management agencies

There is a significant amount of existing background and environmental baseline data available for the Project. This data continues to be collected and reported to the regulators as part of operational controls. Additional data and environmental studies (Section 20.3) will be of technical interest to the federal and state agencies in evaluating a request to expand the current mining operation.

Permits currently held by the HGM may be kept, modified, terminated, or replaced during the expansion process.

1.11 Capital and Operating Costs

The total LOM capital cost is US$500.8 million (excluding net preproduction revenue credit), this is summarized in Table 1-9. The capital cost during the expansion phase is estimated to be US$254.7

million, which includes expansion of the process plant to 4.0 Mtpa, open pit fleet expansion to meet higher throughput, underground development and surface infrastructure. During the commissioning period (through September 30, 2017) a US$69.2 million net revenue credit is generated. The calculation for the net revenue credit is preproduction revenue minus preproduction operating cost. The sustaining capital cost thereafter is US$246.1 million, which is primarily for surface infrastructure, open pit mine equipment replacements and underground development.

Table 1-9: Total Capital Cost Summary (US$000's)

Description	Expansion	Sustaining	Total
Open Pit	59,928	73,068	132,996
Underground	55,361	25,696	81,057
Process Plant	58,881	25,570	84,451
Infrastructure	64,822	119,551	184,373
Total	**238,991**	**243,885**	**482,876**
Contingency	15,725	2,200	17,925
Total Capital Before Revenue Credit	**254,716**	**246,085**	**500,800**
Net Preproduction Revenue Credit	(69,153)	-	(69,153)
Total LoM Net Capex	**185,563**	**246,085**	**431,648**

Source: SRK

<u>Operating Cost</u>

The total LOM operating cost (excluding capitalized operating cost) is US$1,813 million, this is summarized in Table 1-10. Operating costs have been estimated using first principles, derived from supplier quotations, estimated internally by OceanaGold or have been sourced from the Haile 2016 LOM plan.

Table 1-10: Total Operating Cost Summary (US$000's)

Description	Total Cost
Open Pit	771,961
Underground	120,133
Processing	596,491
General & Administration	190,313
Selling/Refining	6,213
Indirect Costs	127,837
Total	**1,812,948**

Source: SRK

1.12 Economic Analysis

The Project consists of a mill expansion from a starter operation operating at 6,300 t/d to a full 11,000 t/d processing capacity by January 2021. The plant is mainly fed by the OP mine which starts production at 11,000 t/d mined rate and subsequently increases production to 40,000 t/d once the mill expansion is completed in 2021. The expanded mill feed is also supplemented with ore from a five-year UG 1,921 t/d high grade operation that begins production in the same year of the expanded mill capacity comes online in 2021.

The Project is expected to produce 2.854 million ounces of gold over a 16-year mine life at a rate of 182 koz Au per year with a RoM All-In Sustaining Cost (AISC) of US$701/oz which can be subdivided into three phases:

1) Starter OP AISC (Oct 2017-2020): US$669/oz Au;
2) UG+OP AISC (2021-2025): US$746/oz Au; and
3) Ending OP AISC (2026-2032 EOM): US$677/oz Au.

The development/expansion capital cost is estimated at US$255 million that would be partially offset by a Net Preproduction Revenue credit of US$69.2 million.

Project metrics using a constant US$1,300 / oz gold price include pre-tax and after-tax NPV 5% values of US$925.3 million and US$770.5 million, respectively. Because the project is valued on a total project basis and not by an incremental analysis of the UG start up and mill expansion, an IRR value is not relevant in this analysis. In terms of sensitivity, the Project is not surprisingly most sensitive to gold grade and price following by operating costs and capital costs. The Project after-tax NPV 5% value changes US$1.50 million per US$1.00 change in the gold price.

1.13 Conclusions and Recommendations

1.13.1 Geology and Mineral Resources

Open Pit

Geologic models will be refined and expanded based largely on core drilling and pit mapping to better understand controls to mineralization which, in turn, will define new drill targets at Haile and in the region. 3D geologic domains will be defined to improve grade estimation. Structural interpretation with improved alteration zonation and geochemistry will be used to develop new drill targets within the upper Persimmon Fork Formation and in the overlying Richtex metasediments. Lithology, alteration and mineralogical data must be integrated with geotechnical, overburden storage and metallurgical evaluations to optimize and realize reserve growth. All targets within the mine permit boundary will be reviewed in light of new geologic models and expanded pit designs.

Underground

SRK makes the following conclusions and recommendations for the Horseshoe underground:

- OceanaGold has completed an industry standard resource definition drilling program to delineate the Horseshoe mineralization sufficient to support the current Mineral Resource estimation;
- The average drill spacing is approximately 25 m x 25 m within the Indicated Mineral Resource and 50 m x 50 m in the Inferred Mineral Resource;
- The exploration work has been accompanied by an industry standard QA/QC program showing good quality analytical results;
- OceanaGold has conducted extensive core logging resulting in a high quality geologic model;
- The results of the drilling, sampling, analytical testing, core logging and geologic interpretation provide good support for an industry standard resource estimation; and
- The results of the Mineral Resource estimation define an Indicated Mineral Resource of 2.71 Mt at an average Au grade of 5.68 g/t containing 0.5 Moz of gold and an additional Inferred Mineral Resource of 1.2 Mt at an average Au grade of 5.0 g/t containing 0.2 Moz of gold.

1.13.2 Status of Exploration; Development and Operations

OceanaGold will continue to expand resources adjacent to open pit and underground reserves in the Haile district by aggressive development drilling and by regional greenfields exploration in the Carolina terrane. Systematic target generation and rationalization supported by mapping, drilling, geochemistry and geophysics is expected to yield new discoveries within 30 km of Haile during the next five years via focused exploration programs. Interpretation of geophysical data and integration with geology and geochemistry will play an important role in exploring under Coastal Plain Sands. It is worth noting that a blind gold deposit has never been discovered in the Carolina terrane. An ‘exploration toolkit’ of diagnostic criteria for Haile-like deposits has been developed to drive exploration for potentially mineable deposits supported by a plethora of historic data and exemplary community relations.

1.13.3 Mining and Reserves

Open Pit

The mine block model, geotechnical stability, pit design, phase design, dump design, production schedule and reserve estimation have been completed to a feasibility study standard. The Project confirms a positive cash flow using only Measured and Indicated Resources for the conversion of reserves using a US$1,300/oz gold price. The mine design supports the style and size of equipment selected for operations. While subject to continual improvement, the mine plan implementation will require qualified staff and the integration of all mining and related disciplines for the successful execution of the Project.

The mine operating and capital costs have been estimated from first principels and operational knowledge from current mine operations. The equipment is sized to meet minimum SMU requirements that support the dilution and mine recovery factors while providing bulk earthwork capability for the expected production rates.

The LoM production schedule includes provision for careful control of potentially acid generating overburden and appropriate material handling costs have been included in the mining cost estimate.

Underground

Longhole stoping is seen as the appropriate mining method for the deposit geometry. The large stope sizes minimize cost and grades are not overly diluted. Mine planning work considered revenue for Au and a cut-off grade (CoG) of 1.5 g/t was used. Stope optimization was completed to identify economic mining areas. A 3D mine design was completed based on the stope optimization results.

The underground mine is accessed via ramp, with the ramp portal located on an open pit bench approximately 80 m below natural surface. Two 5 m raises to surface serve as intake and exhaust raises. The intake raise is outfitted with an emergency egress system.

Tonnage and grades presented in the reserve include dilution and recovery and are benchmarked to other similar operations. Productivities were generated from first principles with inputs from mining contractors, blasting suppliers, and equipment vendors where appropriate. The productivities were also benchmarked to similar operations. Equipment used in this study is standard equipment used worldwide with only standard package/automation features.

A monthly/quarterly/yearly production schedule was generated using iGantt software. The schedule targeted 1,924 t/d.

1.13.4 Mineral Processing and Metallurgical Testing

A significant portion of equipment installed at the Haile process plant was designed conservatively enough (that is, with sufficient additional capacity) to readily accommodate expansion.

The existing Haile facilities have sufficient additional capacity in the major equipment and site footprint to readily allow an expansion to a capacity of 4.0 Mtpa.

Identification of the location of the additional equipment and other design allowances extant in the current plant will be identified during the forthcoming ramp-up phase of the operation. This is expected to allow operation of the plant at above the current nameplate capacity and reduce the additional requirements for the proposed expansion.

No novel, experimental or unproven technologies are used for the Haile process plant or will be used in the proposed expanded plant.

1.13.5 Recovery Methods

There is no effective change to the existing plant recovery method proposed for the plant expansion compared to the current circuit configuration.

The processing plant is operating and is in the final stages of commissioning and entering the ramp up phase. The process selected is conventional and appropriate and in this context readily expandable to suit the accelerated processing rate.

The existing Haile facilities have an ample site footprint to expand to a capacity of 4.0 Mtpa.

This would bring the plant’s major components close to their maximum capabilities.

1.13.6 Project Infrastructure

The surface infrastructure required for the increase in open pit production as per the mine plan is straightforward as it is aligned with the current surface infrastructure requirements. The primary changes are increases in size to the main waste storage facilities such as the TSF and PAG OSA’s. Water management will remain a challenge and it is proposed to develop a LOM diversion channel for the HGMC to reduce the risk with management of pipe diversions on pit benches.

The surface infrastructure required for the development of the underground Horseshoe deposit is straightforward and relatively minor. Consideration should be given into whether the crusher is required for support of the CRF plant.

1.13.7 Environmental Studies and Permitting

There is a significant amount of existing background and environmental baseline data available for the Project. This data continues to be collected and reported to the regulators as part of operational controls. Additional data and environmental studies (Section 20.3) will be of technical interest to the federal and state agencies in evaluating a request to expand the current mining operation.

Permits currently held by the HGM may be kept, modified, terminated, or replaced during the expansion process.

OceanaGold will work closely with all key stakeholders to ensure that the permitting of the mine expansion meets all federal and state requirements.

1.13.8 Economic Analysis

The Project consists of a mill expansion from a starter operation operating at 6,300 t/d to a full 11,000 t/d processing capacity by January 2021. The plant is mainly fed by the OP mine which starts production at 11,000 t/d mined rate and subsequently increases production to 40,000 t/d once the mill expansion is completed in 2021. The expanded mill feed is also supplemented with ore from a five-year UG 1,921 t/d high grade operation that begins production in the same year that the expanded mill capacity comes online in 2021.

Capital and operating costs are broadly in line with current market expectations and benchmark data from other OceanaGold operations. The Project is expected to produce 2.854 million ounces of gold over a 16-year mine life at a rate of 182 koz Au per year with a RoM AISC of US$701/oz. Project metrics using a constant US$1,300/oz gold price include pre-tax and after-tax NPV 5% values of US$925.3 million and US$770.5 million, respectively.

The expanded process plant capital and operating cost estimates will be refined during the forthcoming phases of definitive engineering study and detailed design.

2 Introduction

2.1 Terms of Reference and Purpose of the Report

This National Instrumented 43-101 (NI 43-101) Technical Report (Technical Report) was prepared as for OceanaGold Corporation (OceanaGold) to a feasibility level by SRK Consulting (U.S.), Inc. (SRK) on the Haile Gold Mine (Haile or Project). This report includes both open pit and underground mining components and a single economic analysis based on open pit and underground reserves.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in SRK's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by OceanaGold subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits OceanaGold to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101 Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with OceanaGold. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report is issued.

This report provides Mineral Resource and Mineral Reserve estimates, and a classification of resources and reserves prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 (CIM, 2014).

2.2 Qualifications of Consultants

The consultants preparing this technical report are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, underground mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

SRK consultants employed in the preparation of this Technical Report have no beneficial interest in OceanaGold. The SRK consultants are not insiders, associates, or affiliates of OceanaGold. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between OceanaGold and SRK. The SRK consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, and are members in good standing of appropriate professional institutions. QP certificates of authors are provided in Appendix A. The QPs are responsible for specific sections as follows:

- David Carr, BEng Metallurgical (Hons), OceanaGold Chief Metallurgist is the QP responsible for mineral processing, Sections 13, 13.1, 13.1.1, 13.1.2, 13.1.3, 13.2, 17, 17.1, 17.2, 17.2.1, 17.2.2, 17.2.3, 17.2.4, 17.2.5, 17.2.6, 17.2.7, 17.2.8, 17.3, 17.4, 21.2, 21.2.1, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.

- Bruce Van Brunt, MSc., B.A, Fellow AusIMM, (OceanaGold Technical Services Manager), is the QP responsible for open pit Mineral Resources, Sections 4.1, 4.2, 5.1, 5.2, 5.3, 5.4, 5.5, 14.1.1, 14.1.2, 14.1.3, 14.1.4, 14.1.5, 14.1.6, 14.1.7, 14.1.8, 14.1.9, 14.1.10, 14.1.11, 14.1.12, 14.1.13, 14.1.14, 14.1.15, 14.1.16, 16.1.1, 16.1.3, 16.1.5, 16.1.7, 18.5, 18.6, 18.7, 18.8, 18.9, 20, 20.1, 20.2, 20.3, 21.1, 21.1.1, 23, 23.1, 23.2, 23.3, co-author 14, 14.3, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- John Jory, MSc Mineral Exploration, BS Geology, CPG, AIPG, (OceanaGold Director of Exploration, Haile), is the QP responsible for the sections on history, geology, mineralization, drilling, sampling and analysis in Sections 6, 7.1, 7.2, 7.2.1, 7.2.2, 7.2.3, 8, 8.1, 8.2, 9.1, 9.2, 9.3, 9.3.1, 9.3.2, 10.1, 10.2, 10.3, 10.4, 11.1, 11.1.1, 11.1.2, 11.2, 11.3, 11.4, 11.4.1, 11.4.2, 11.4.3, 11.4.4, 11.5, 12, 12.1, 12.1.1, 12.1.2, 12.2, 12.2.1, 12.3, 12.3.1, 12.3.2, 12.3.3, 12.4, 12.4.1, 12.4.2, 12.4.3, 12.5, 12.5.1, 12.5.2, 12.5.3, 12.6, 12.6.1, 12.6.2, 12.6.3, 12.6.4, 12.7, 12.7.1, 12.7.2, 12.7.3, 12.7.4, 12.7.5, 12.7.6 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Jay Newton Janney-Moore, PE, (NewFields Senior Engineer IV), is the QP responsible for tailing and overburden storage, Sections 18.1, 18.2, 18.2.1, 18.3, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Paul Howe, BSc., MAppSc. Hydrogeology and Groundwater Management, (CDM Smith Principal Hydrogeologist) is the QP responsible for hydrogeology, Sections 16.1.8, 16.2.3, 18.4, co-author 16.1.2, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Joanna Poeck, BEng Mining, SME-RM, MMSAQP, (SRK Senior Consultant, Mining Engineer), is the QP responsible for underground Mineral Reserves, Sections 2.1, 2.2, 2.3, 2.4, 2.5, 2.6, 3, 15, 15.2, 15.2.1, 15.2.2, 15.2.3, 16, 16.2, 16.2.1, 16.2.5, 16.2.6, 16.2.8, 24, 27, 28, 28.1, 28.2, 28.3, 28.4, co-authored 15.3, 16.3, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Jeff Osborn, BEng Mining, MMSAQP, (SRK Principal Consultant, Mining Engineer), is the QP responsible for underground mining and infrastructure, Sections 16.2.7, 16.2.9, 16.2.11, 18.7.1, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Derek Kinakin, MSc, P.Geo, (BGC Senior Engineering Geologist), is the QP responsible for open pit geotechnical work, co-authoring Section 16.1.2 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Grant Malensek, MEng, PEng/PGeo, (SRK Principal Consultant, Mineral Economics), is the QP responsible for technical-economics Sections 19, 19.1, 22, 22.1, 22.2, 22.2.1, 22.2.2, 22.2.3, 22.2.4, 22.2.5, 22.3, 22.4, 22.4.1, 22.4.2, 22.4.3, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- David Bird, MSc., PG, RM-SME, (SRK Principal Consultant, Geochemistry), is the QP responsible for underground geochemistry, Section 16.2.4, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Bret C. Swanson, BEng Mining, MAusIMM, MMSAQP, (SRK Practice Leader/Principal Consultant, Mining Engineer), is the QP responsible for open pit mining and Mineral Reserves, Sections 15.1, 15.1.1, 15.1.2, 15.1.3, 16.1, 16.1.4, 16.1.6, co-author 15.3, 16.3, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.

- John Tinucci, PhD, PE, (SRK President/Practice Leader/Principal Consultant, Geotechnical Engineer), is the QP responsible for underground Mineral Resource estimates, Section 16.2.2 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Bart A. Stryhas, PhD, CPG (SRK Principal Associate Resource Geologist), is the QP responsible for underground Mineral Resource estimates, Sections 14.2, 14.2.1, 14.2.2, 14.2.3, 14.2.4, 14.2.5, 14.2.6, 14.2.7, 14.2.8, 14.2.9, 14.2.10, 14.2.11, 14.2.12, 14.2.13, 14.2.14, co-author 14, 14.3 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Brian S. Prosser, PE, SRK Principal Consultant, is the QP responsible for ventilation, Section 16.2.10 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.

2.3 Details of Inspection

Site visits conducted by QPs are summarized in Table 2-1.

Table 2-1: Site Visit Participants

Personnel	Company	Expertise	Date(s) of Visit	Details of Inspection
Derek Kinakin	BGC	Geotechnical	February 6 to 8, 2017	Review of existing pit slopes and examination geotechnical drill core.
David Carr	OceanaGold	Metallurgy	March 28, 2017 through July 18, 2017	Plant Commissioning support and plant investigations Plant Commissioning and ramp-up of operations
Jay Newton Janney-Moore	NewFields	Geotechnical/ Infrastructure	March 15 to 17, 2017	Inspection of the Duckwood TSF, PAG OSA, and geomembrane lined ponds
Paul Howe	CDM Smith	Hydrogeology	November 22 to December 6, 2017	Review of hydrogeological response to dewatering and depressurisation
Jeff Osborn	SRK	Mining/ Infrastructure	February 6 and 7, 2017	Site tour of current mining and infrastructure
Joanna Poeck	SRK	UG Mining/ Infrastructure	February 6 and 7, 2017	Site tour of current mining and infrastructure
Bret Swanson	SRK	OP Mining	May 9 and 10, 2017	Site tour of current mining
John Tinucci	SRK	Geotechnical	November 29 to December 1, 2016	Site tour of current mining, geotechnical drilling activities, and examination of core
Bart Stryhas	SRK	Geology/Resources	October 10 to 12, 2016	Site tour including core logging facility, examination of core, and analytical laboratory.

Bruce van Brunt and John Jory are based in South Carolina and are on-site regularly.

2.4 Sources of Information

This report is based in part on internal Company technical reports, previous feasibility studies, maps, published government reports, Company letters and memoranda, and public information as cited throughout this report and listed in the References Section 27.

2.5 Effective Date

The effective date of this report is January 1, 2017.

2.6 Units of Measure

The Metric System for weights and units has been used throughout this report. Tonnes are reported in metric tonnes of 1,000 kg. Gold is reported in grams and troy ounces, where applicable (1 Troy ounce = 31.1035 grams). All currency is in U.S. dollars (US$) unless otherwise stated.

3 Reliance on Other Experts

The Consultant's opinion contained herein is based on information provided to the Consultants by OceanaGold throughout the course of the investigations. SRK has relied upon OceanaGold and the work of other consultants in the project areas in support of this Technical Report.

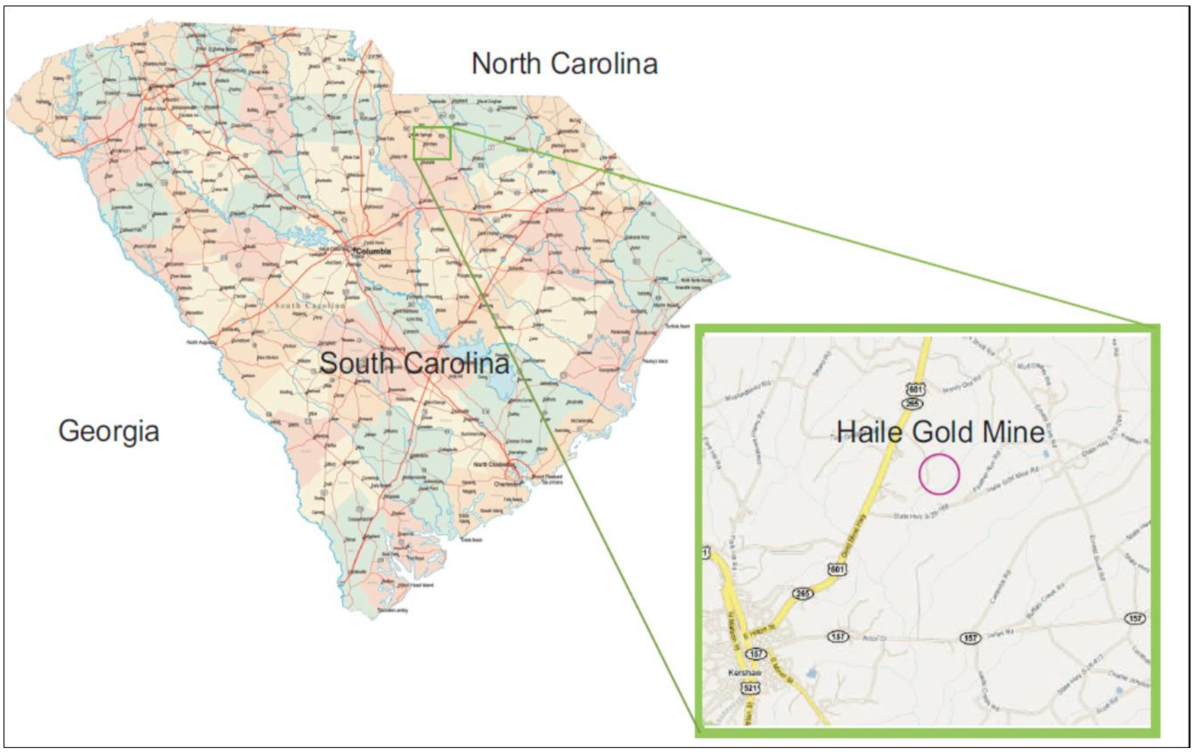
The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

SRK has relied upon OceanaGold for information regarding the surface land ownership/agreements as well as the mineral titles and their validity. Land titles and mineral rights for the project have not been independently reviewed by SRK and SRK did not seek an independent legal opinion for these items.

4 Property Description and Location

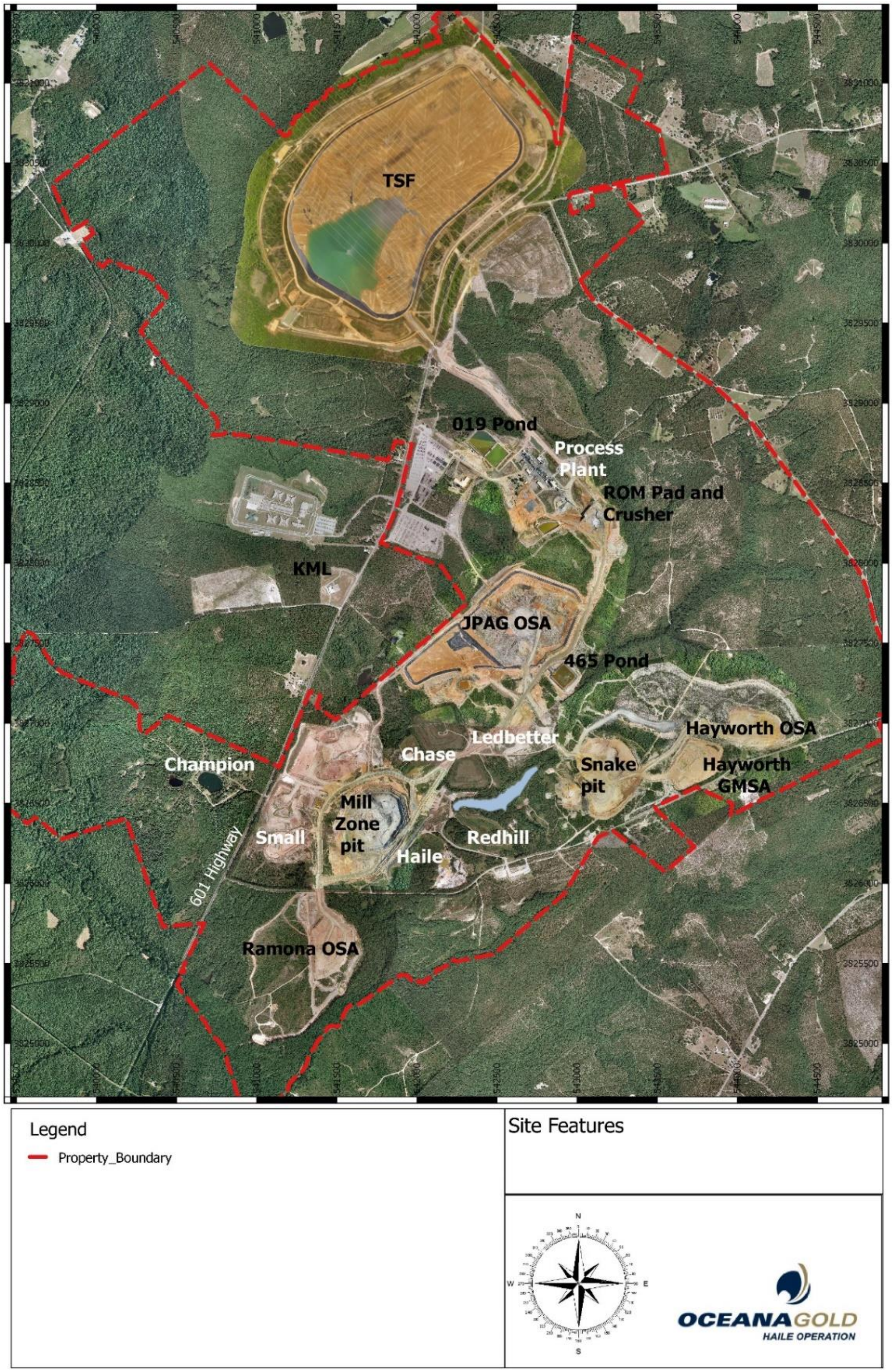
4.1 Property Location

The Haile property is located 5 km northeast of Kershaw in southern Lancaster County, South Carolina, USA, in the north-central part of the state, as shown in Figure 4-1. Haile is 27 km southeast of Lancaster, the county seat, and is 80 km northeast of Columbia, the state capital. The approximate geographic center of the property is at 34° 34' 46" N latitude and 80° 32' 37" W longitude. The mineralized zones at Haile lie within an area extending from UTM NAD83 zone 17N coordinates 540000E to 544000E and 3825500N to 3827500N. Figure 4-2 shows a site map of Haile.



Source: State-Maps.org and Google Maps, 2014

Figure 4-1: General Location Map of the Haile Gold Mine



Source: OceanaGold

Figure 4-2: Site Map of the Haile Gold Mine

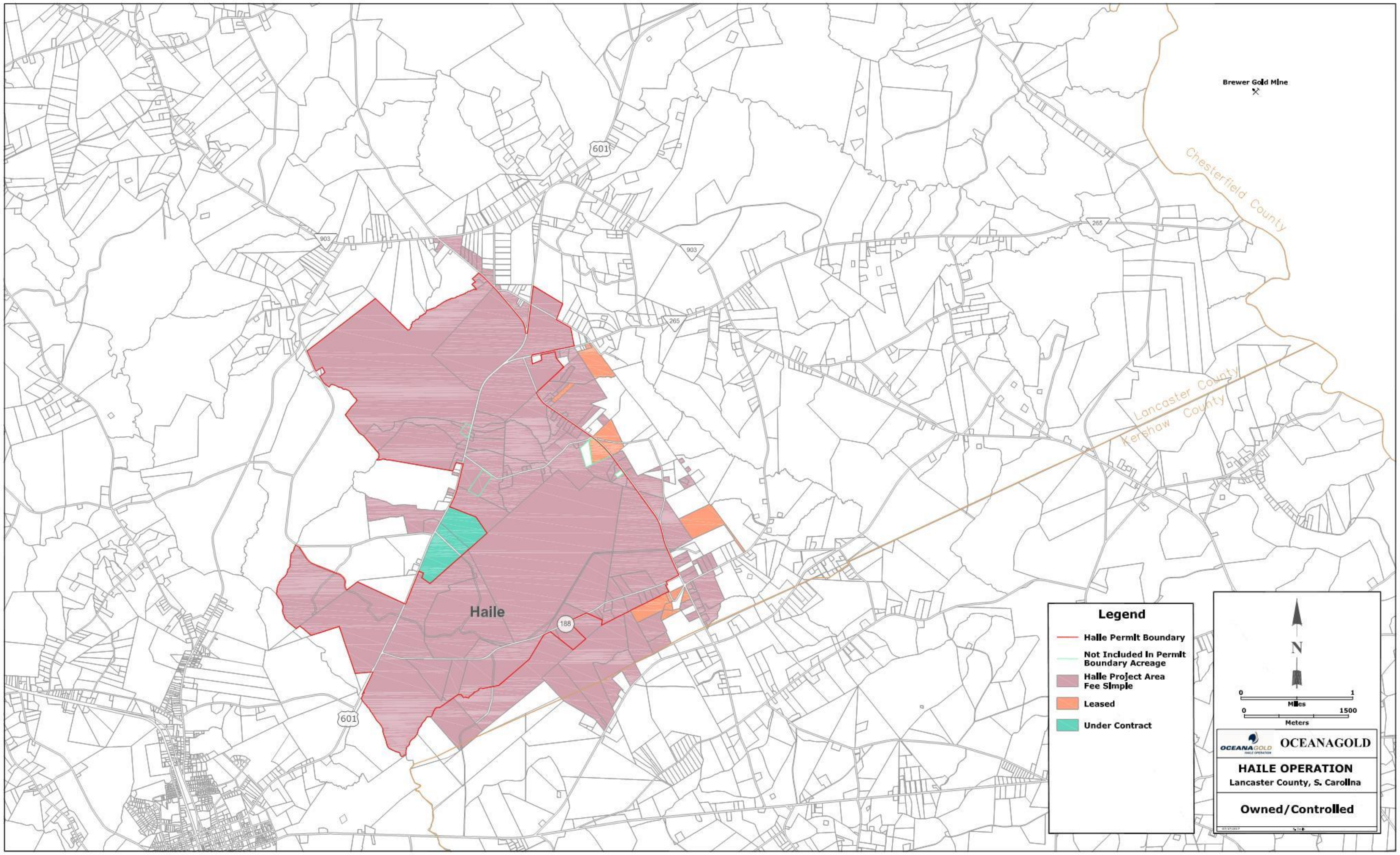
4.2 Ownership

Haile Gold Mine Inc. (HGM) is a wholly owned subsidiary of OceanaGold Corporation (OceanaGold). References in this document to OceanaGold refer to the parent company together with its subsidiaries, including HGM and Romarco Minerals Inc.

As of June 30, 2017, HGM owns approximately 2,440 hectares of land in total. 1,842 hectares of this total are in the mine permit area, 149 hectares are for conservancy purposes.

HGM owns additional land that is not associated directly with the project.

Figure 4-3 shows the Land Tenure map. It includes Haile Project Area Fee Simple, Leased, and Under Contract land for the Haile area.



Source: OceanaGold

Figure 4-3: Land Tenure Map

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

Haile is accessible by car or truck by taking U.S. Highway 601 northeast from the town of Kershaw for 5 km to the mine gate on Snowy Owl Road. The major international airport at Charlotte, North Carolina, is an 80-minute drive from the mine.

5.2 Climate

The Haile area of South Carolina has a sub-tropical climate. Summers are hot and humid with daytime temperatures averaging 29°C to 35°C. Winters are mild and wet, and overnight temperatures can be below freezing (0°C). Average annual precipitation is 1,270 mm while annual evaporation is estimated at 760 mm. Precipitation is abundant throughout the year with January, March and July being the wettest months. Snowfall is insignificant and averages less than 80 mm per year. South Carolina averages 50 days of thunderstorm activity and 14 tornadoes per year. The mine operating season is considered to be year-round.

5.3 Local Resources and Infrastructure

Local resources (labor force, manufacturing, housing, utilities, emergency services, etc.) and infrastructure are in place and are widely utilized for the operation of Haile. Several small and modest-sized communities exist around Haile.

Power is available in the area via an existing 44 kV transmission grid with Duke Energy and a 69 kV transmission grid with Lynches River. The company utilizes both grids.

5.4 Physiography

Haile and its surroundings occur within the Sand Hills sub-province of the Piedmont physiographic province of the southeastern United States. This province trends from southwest to northeast and is bound by the Coastal Plain to the southeast and the southern Appalachian Mountains to the northwest. Gentle topography and rolling hills, dense networks of stream drainages, and white sand to red-brown saprolitic soils characterize the province.

The elevation of the property ranges from 120 m to 170 m above mean sea level. Topography is dissected by the perennial, southwest-flowing HGMC and by its intermittent, southeast and northwest-flowing tributaries. HGMC enters the southeast-flowing Little Lynches River 1.6 km southwest of the mine site. Gradients within the drainages are gentle to moderate (9 to 13%) and slopes above the drainages are gentle to nearly flat (less than 1%). The property is heavily wooded by pine and hardwood forests. Pine timber harvesting occurs frequently in and around the property area as each harvestable tract matures.

5.5 Infrastructure Availability and Sources

There are large industrial population centers near the Project site, Equipment and sources of both logistical and professional expertise can be obtained from the major cities of Charlotte, North Carolina and Columbia, South Carolina., which are both within one-hour travel of the mine. More than one large industrial contractor is within close proximity to the site and can provide a skilled workforce for the project. There is adequate labor for operations.

6 History

Gold was discovered in 1827 near Haile by Colonel Benjamin Haile, Jr. in gravels of Ledbetter Creek (now HGMC). This led to placer mining and prospecting until 1829, when lode deposits at the Haile-Bumalo pit site were found. Surface pit and underground work continued at the Haile-Bumalo site for many years. In 1837 a five-stamp mill was built (Newton et al.., 1940). Gold production and pyrite-sulfur mining for gunpowder continued through the Civil War from 1861 to 1865. General Sherman's Union troops invaded the area and burned down the operations near the war's end.

In 1882 a sixty-five-stamp mill was constructed by E.G. Spilsbury and operated continuously until a fatal boiler explosion killed the mine manager in 1908. During that time, Adolph Thies developed the Thies barrel chlorination extraction process and improved gold recovery from Haile sulfide ores (Pardee and Park, 1948). During the 26-year operation period, mining grew to include the Blauvelt, Bequelin, New Bequelin, and Chase Hill areas. From 1907 to 1913 an attempt to operate a cyanide plant to extract gold from mine tailings was unsuccessful. Pyrite used to produce sulfuric acid was mined at Haile from 1914 to 1918 (Newton et al.., 1940).

From mid-1937 to 1942 larger-scale mining was undertaken by the Haile Gold Mines Company. The property then consisted of owned or leased ground totaling about 1,335 hectares (ha) (Newton et al., 1940). Most of the main pits were mined to the 46-m level with some underground operations at Haile-Bumalo reaching the 106-m level (Pardee and Park, 1948). The Red Hill Deposit was discovered by crude induced polarization techniques next to the Friday pyrite diggings (Newton et al., 1940). This fairly large operation was shut down by presidential decree in 1942 because of World War II. By this time, Haile had produced over US$6.4 million worth of gold (in 1940 dollars) (Newton et al., 1940).

Starting in 1951 the Mineral Mining Company (Kershaw, South Carolina) mined Mineralite® from sericite-rich pits around Haile. This industrial product is a mixture of sericite, kaolinite, quartz, and feldspar and is used in manufacturing insulators and paint base. Mineralite mining ended in 1991.

In 1966 Earl Jones conducted exploration work in the area and eventually interested Cyprus Exploration Company (Cyprus) in the project. Cyprus worked Haile from 1973 to 1977. Numerous companies explored the Haile area in the 1970s and 1980s, including Amselco, Amax, Nicor, Callaghan Mining, Westmont, Asarco, Newmont, Superior Oil, Corona, Cominco, American Copper and Nickel, Kennecott, and Hemlo.

The 1980s heralded the first successful modern exploration and production at Haile. Piedmont Land and Exploration Company (later Piedmont Mining Company) explored Haile and surrounding properties from 1981 to 1985. Piedmont drilled 67 core holes and 1,215 reverse circulation holes on the property and greatly expanded the footprint of the Haile deposits. Piedmont mined the Haile deposits from 1985 to 1992, and produced 85,000 ounces of gold from open pit heap leach operations in oxide and transitional ores. New areas mined by Piedmont included the Gault Pit (next to Blauvelt), the 601 pits (by the US 601 highway), and the Champion Pit. Piedmont expanded the Chase Hill and Red Hill pits and combined the Haile-Bumalo zone into one pit. Piedmont also discovered the large Snake deposit sulfide gold resource and mined its small oxide cap. Piedmont extracted gold ores from a mineralized trend 1.6 km long, from east to west. Historical gold production at Haile is estimated at 360,000 ounces (Speer and Madry, 1993, Maddry and Kilbey, 1995).

In June 1991 Amax signed an agreement to evaluate Haile to determine if it should enter into a joint venture. During the evaluation period, core drilling stepped north of the Haile-Bumalo area and discovered the new sulfide resource at the Mill zone under the old 1940s mill. Amax and Piedmont entered into a joint venture agreement and established the Haile Mining Company (HMC) in May 1992.

From 1992 to 1994, HMC completed a program of exploration and development drilling, property evaluation, Mineral Resource estimation, and technical report preparation. During this period, the large Ledbetter resource zone was discovered under a mine haul road. At the end of the HMC program in 1994, the gold reserve was stated as 780,000 ounces of gold contained within 7.9 million tonnes at an average gold grade of 3.05 g/t. A qualified person has not done sufficient work to classify the historical estimate as Mineral Resources or Mineral Reserves. HGM is not treating the historical estimate as Mineral Reserves. Because of unfavorable economic conditions at the time, Amax did not proceed with mining, and began a reclamation program to mitigate acid rock drainage (ARD) conditions at the site.

Kinross acquired Amax in 1998, assumed Amax's portion of the Haile joint venture, and later purchased Piedmont's interest. Because Haile was a low priority compared to larger and more profitable projects, Kinross decided not to reopen the mine and continued the reclamation and closure program. Reclamation and closure proceeded through to 2015 when Haile Operations commenced under Romarco.

Romarco acquired Haile from Kinross in October 2007 and began a confirmation drilling program in late 2007. Romarco completed the confirmation drill program in early 2008 and began infill and exploration drilling focused around the Ledbetter resource. Drilling accelerated in early 2009 with a major reverse circulation infill drilling program that continued into 2013. Condemnation drilling by Romarco for mine facilities commenced in September 2009. Drilling east of the Snake deposit discovered the high-grade Horseshoe deposit in 2010 and required the planned tails storage facility to be relocated 3 to 4 km northwest of the mine. Geotechnical drilling was initiated in September 2009 for pit slope designs. The final hole at Ledbetter discovered a deeper northwest extension in 2010 that was named Mustang. Drilling between the Red Hill and Horseshoe areas had identified large zones of low grade material that led to the late 2011 discovery of the deep Palomino prospect. Due to low gold prices and mine permitting, Haile exploration drilling was suspended during 2013 and 2014.

Romarco submitted a feasibility study for Haile in February 2011. Drill hole data available as of November 17, 2011, were used in the March 2012 Mineral Resource estimate. Romarco completed a large portion of detailed engineering and permitting for the project in 2011 and 2012. In November 2014, an updated feasibility study was completed after receiving the necessary permits. In April 2015 construction of the project began by Romarco and mining commenced in the Mill Zone pit.

OceanaGold Corporation acquired Romarco Minerals Inc. in October 2015 and became owner and operator of Haile. Project construction during 2015 and 2016 included a new Carbon-In-Leach (CIL)-flotation process plant, power upgrades, a lined PAG overburden storage area, and a tails storage facility. The first gold pour at the plant was in January 2017.

7 Geological Setting and Mineralization

7.1 Regional Geology

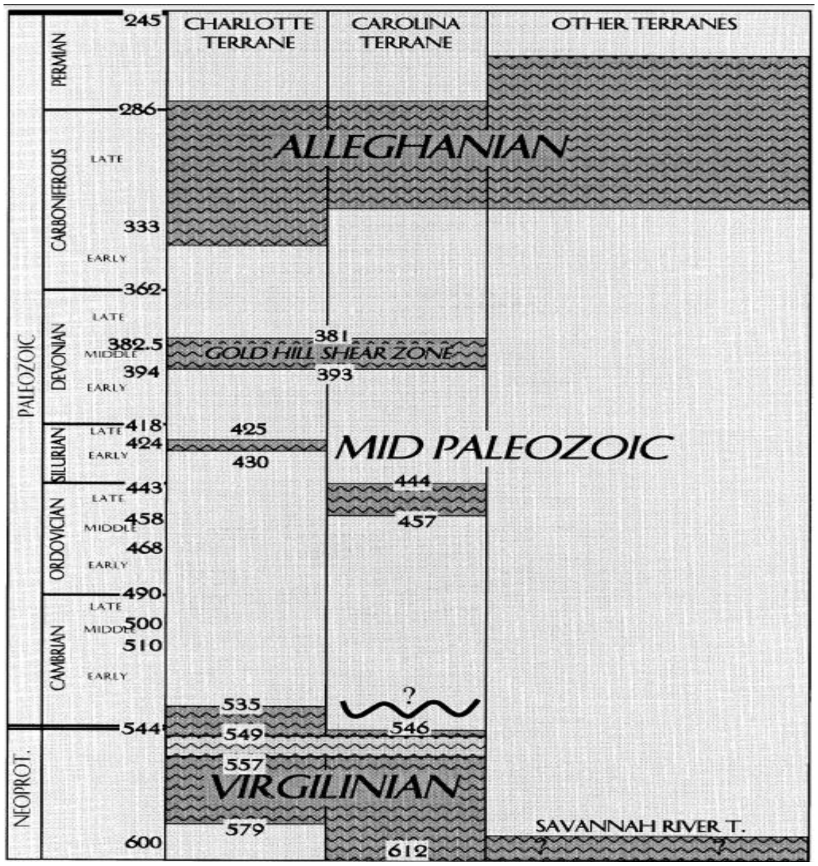
The largest gold deposits in the southeastern USA are located in the Carolina terrane (Secor et al., 1983) in the north-central portion of South Carolina. The Carolina terrane extends 600 km from eastern Virginia to central Georgia, and is up to 140 km wide in North Carolina. The Carolina terrane is also known as the Carolina Slate Belt. The Carolina terrane is composed predominantly of Neoproterozoic to Cambrian metaigneous and metasedimentary rocks. The Carolina terrane has a prominent flexure in central South Carolina near Haile. Structural trends southwest of this area are east–northeasterly, whereas those northeast of the flexure are northeasterly (Hibbard et al., 2002).

The Carolina terrane consists of the Carolina terrane, the Charlotte terrane, the Augusta-Dreher Shoals terrane and the Kings Mountain terrane (Hatcher et al., 2007 and Hibbard et al., 2007). These exotic, volcanic arcs formed adjacent to the African continent and were accreted to the North American craton during the Late Ordivician–Silurian (Hibbard et al., 2010). Gold mineralization at Haile (~549 Ma, Mobely et al., 2014) occurred during suturing of the Carolina and Charlotte arcs coincident with emplacement of the Longtown and Little Mountain stitching plutons (Barker et al., 1998).

Four general tectonothermal periods are recorded in the Carolina terrane, including (Hibbard et al., 2002) as shown in Figure 7-1 and described as follows:

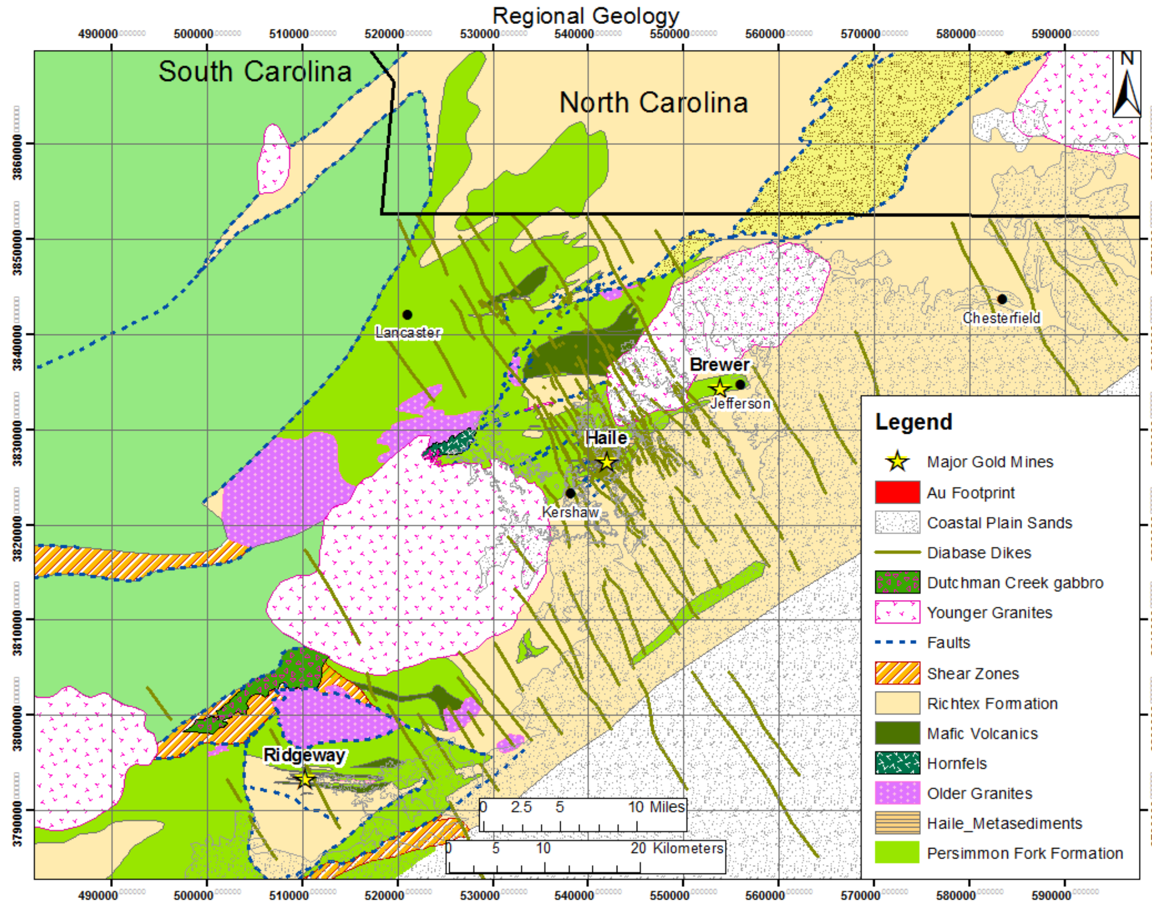
1. Late Neoproterozoic to Early Cambrian *Virgilina* events (617 to 544 Ma): folding, foliation and faulting with granite plutonism;
2. Late Ordovician–Silurian *Taconian* events (457 to 425 Ma): upright folding with a penetrative
3. Axial planar cleavage accompanied by greenschist facies metamorphism;
4. Devonian events of the Gold Hill-Silver Hill dextral shear zone (393 to 381 Ma) that juxtaposes the Carolina and Charlotte terranes; and
5. Late Paleozoic *Alleghanian* events (333 to 286 Ma): ductile mylonitic shear zones (e.g., Hyco, Modoc shears) (Hibbard et al., 1998) with orogenic quartz veins and greenschist to amphibolite facies metamorphism and granite plutonism.

Post-tectonic granites intruded into the Carolina superterrane at the end of the Alleghanian orogeny. The Alleghanian-aged Liberty Hill and Pageland granite plutons are exposed west and north of Haile (Figure 7-3). Intermediate to mafic dikes of Carboniferous age and prominent northwest-trending Mesozoic diabase dikes intrude the Carolina terrane. Erosion and weathering with saprolite formation up to 40 m deep have occurred in the region since the Mesozoic due to a sub-tropical, humid paleo-environment. Regional submersion during the Cretaceous resulted in the deposition of a southeastward-thickening apron of kaolinitic sands and clay above the saprolite. Continental uplift and regression of the Atlantic Ocean have led to continued erosion and dominantly southeast-flowing drainages.



Source: Hibbard et al., 2002

Figure 7-1: Time distribution of major orogenic events in the Carolinas



Source: OceanaGold

Figure 7-2: District geology of north-central South Carolina (UTM NAD83 Z17N)

Volcaniclastic sedimentation is characteristic of convergent plate margins in marine forearc sequences and in marine and non-marine back arcs and grabens. The forearc region between the volcanic arc and the down-going subducting crustal slab, above which lies the trench, can be up to 300 km wide and can form a large forearc basin (Fisher and Schmincke, 1994). Sedimentary environments along the shoreline include beach-shelf-slope-rise with fan-deltas, deltas, submarine canyons and submarine fans. The volcaniclastic component within the sedimentary fill depends upon intensity of volcanic activity and the volume of debris that enters this environment. Within the forearc basin itself, sediments are largely turbidite-dominated, and clastics are epiclastic volcanic and reworked pyroclastites and hydroclastites. Pyroclastic and hydroclastic materials dominate during episodes of volcanism, whereas epiclastics dominate between volcanic episodes.

The largest gold deposits in South Carolina are the Haile (4.4 Moz resource), Ridgeway (1.44 Moz resource) and Brewer (0.2 Moz resource) deposits (Foley and Ayuso, 2012). They are broadly aligned SW-NE and occur at or near the strongly deformed contact between metamorphosed volcanic and sedimentary rocks of Neoproterozoic to Early Cambrian age. Deformation is dominantly ductile with EW to ENE fold axes and isoclinal folds overprinted by shearing and brecciation. The gold deposits are proximal to large granite plutons of variable ages within the regional SW-NE tectonic fabric.

The Brewer mine is located 12 km northeast of Haile and the Ridgeway mine is located 50 km southwest of Haile. Haile and Brewer are hosted in the upper Persimmon Fork Formation in sedimentary and volcanic rocks respectively. Ridgeway is hosted in sheared metasediments of the basal Richtex Formation. Haile is classified as a sediment-hosted intrusion-related disseminated gold deposit with proximal quartz-sericite-pyrite alteration and distal sericite-chlorite alteration as shown in Table 7-1. Ridgeway is geologically similar to Haile in that it is dominantly sediment-hosted with some mineralized volcanic rocks. EW to ENE structural controls and complex folding characterize the Haile and Ridgeway deposits. Brewer is a high sulphidation (pyrite-enargite-chalcopyrite-topaz) volcanic-hosted breccia pipe overprinted by argillic alteration (pyrophyllite-andalusite).

Table 7-1: Geological summary of major gold deposits of SE USA

Deposit	Type	Host rock	Alteration	Resource Moz Au	Au age Ma
Haile	Sediment-hosted intrusion-related	Persimmon Fork metasediments	silica-pyrite-sericite	3.46	549
Ridgeway	Sediment-hosted intrusion-related	Richtex metasediments	silica-pyrite-sericite	1.44	553
Brewer	High sulphidation	Persimmon Fork metavolcanics	Pyrite-enargite-chalcopyrite	0.2	550
Barite Hill	VMS	Persimmon Fork metavolcanics, metasediments	silica-barite-sericite	0.06	566

Source: OceanaGold (Ayuso and Foley, 2012)

7.2 Local Geology

The geologic history of the Haile area includes several major events, as listed below as follows:

Late Pre-Cambrian to Early Cambrian (580 to 530 million years ago)

- Carolina Terrane formed as part of a subduction zone-oceanic island arc complex (Secor, et al., 1983).
- Persimmon Fork Formation consists of andesitic metavolcanic and tuffaceous rocks.
- The sheared upper laminated unit is the primary host rock for Haile mineralization.
- Richtex Formation consists of mudstones and siltstones deposited conformably on the Persimmon Fork metavolcanics (Secor and Wagener, 1968).
- Transition from volcanism to basinal sedimentation ~550 mega-annum (Ma) (Persimmon Fork-Richtex boundary)
- Gold mineralization at Haile dated at ~549 Ma (Mobley, et al., 2014).

Carboniferous (320 to 290 million years ago)

- Continental collision forms the supercontinent of Pangaea and the Alleghenian orogeny in the
- Appalachian Mountains (Hibbard et al., 2002, Hatcher et al., 2007) and emplaced granite plutons
- Liberty Hill granite: 293 ± 15 million years old (Fullagar and Butler, 1979)
- Pageland granite: 296 ± 5 million years old (Fullagar and Butler, 1979)

Triassic - Early Jurassic (250 to 200 million years ago)

- Pangaea begins rifting apart forming rift basins.

- Mafic diabase dikes intrude the Carolina terrane as highly magnetic NW-SE-trending anomalies

Late Cretaceous to Present (100 million years ago)

- Coastal Plain Sands deposited over much of the southeast USA

7.2.1 Lithology

The following rock units are described in chronostratigraphic order from oldest to youngest. Stratigraphy at Haile is described from mapping and core drilling over a thickness of about one km.

The volcanic and interbedded epiclastic rocks of the Haile area are assigned to the approximately three km thick Persimmon Fork Formation that formed 555 to 551 Ma (Hibbard et al., 2002). The Richtex Formation comformably overlies the Persimmon Fork and consists of approximately three km of thin-bedded siltstone, argillite, conglomerate, sandstone and greywacke deposited in a submarine slope environment (Maher et al., 1991, Hibbard et al., 2002). The Persimmon Fork-Richtex boundary marks the ~550 Ma change from volcanic-dominated arc terrane to basinal sedimentary facies.

Recent stratigraphic reinterpretation has reassigned the metasedimentary package at Haile from the Richtex Formation to the uppermost section of the Persimmon Fork Formation. This is supported by fining upward sedimentary cycles, gradational contacts, rapid facies changes, tuffaceous interbeds, and the common occurrence of 1-3% plagioclase crystals in volcaniclastic units. Local pepperite beds tens of meters thick consist of alternating 15 cm to 3 m bands of laminated siltstone and crystal-poor felsic tuff. The conformable ENE-trending contact between the Persimmon Fork and the overlying Richtex Formation is located about 0.5 km south of Haile. Reinterpretation of stratigraphy at Haile simplifies the structural model with a folded volcanic-sedimentary package that is not complicated by overturning or regional thrusting of older rocks over younger rocks.

Persimmon Fork Formation

The Neoproterozoic Persimmon Fork Formation at Haile consists of greywacke, conglomerate, sandstone, siltstone and laminated siltstone grading upward into volcanic tuffs and volcaniclastic rocks (Secor and Wagener, 1968, Secor and Snoke, 1978; Secor et al., 1986). Sedimentary interbeds are common in the volcanic-dominated upper portion of the Persimmon Fork. The sedimentary rocks are conformably overlain by volcanic rocks, and contacts are often gradational. The volcanic rocks are light gray to pale green with blocky weathering patterns that contrast with the platy metasediments. Distinctive subhedral plagioclase crystals 1- to 2-mm in size make up 2 to 10% of the groundmass in the fine-grained crystal tuffs in lower sections of the metavolcanics. Albite and quartz are dominant with lesser muscovite, biotite, calcite, and chlorite. Upper sections of the Persimmon Fork contain poorly sorted, rounded to angular volcanic clasts in lithic tuffs. Volcaniclastic units are commonly interbedded with the tuffs and indicate variable volcanic and detrital inflows over time.

The Persimmon Fork is strongly foliated and is metamorphosed to greenschist facies. Mineralized metavolcanics are silificied and pyritic, but are less extensively mineralized than the underlying metasediments. U-Pb ages from zircons have yielded crystallization ages of 553 +2 Ma (Ayuso et al., 2005).

Richtex Formation

The conformable contact between the Persimmon Fork and Richtex Formations at Haile is defined by the transition from volcanic-dominated rocks with sedimentary interbeds to clastic sedimentary rocks.

The ENE-trending contact is gradational and is located about 0.5 km southeast of Haile). The Richtex Formation consists of thin-bedded siltstone, argillite, conglomerate, sandstone and greywacke about over 3-km thick. The lower portion of the Richtex Formation contains several metavolcanic units including mafic tuff and amydaloidal basalt (Secor and Wagener, 1968). Rocks strike ENE and dip moderately southeast. The Richtex Formation is largely covered by Coastal Plain Sands southeast of Haile.

Lamprophyre Dikes

Lamprophyre (or alkaline) dikes intrude the metasediment and metavolcanic units. Lamprophyres are dark green and fine-grained with porphyritic, spheroidal and mottled textures. The dikes contain biotite and plagioclase with chlorite and calcite. The dikes are mostly ENE-trending and range in thickness from 10 cm to 2 m. 40 Ar/39 Ar dates from biotite samples collected from lamprophyre dikes crosscutting the Haile metasediments yielded Pennsylvanian ages at ~311 Ma, coincident with the Dutchman Creek Gabbro (Mobley et al., 2014).

Diabase Dikes

The diabase dikes are dark gray, dense, medium-grained and magnetic and cut all other units except the Coastal Plain Sands. Dominant minerals are plagioclase and pyroxene with minor chlorite and calcite. The diabase dikes often have chilled margins with calcite flooding. The dikes strike northwest and generally have steep dips that range from 60 to 90 degrees. Diabase dikes typically range in thickness from 1 to 10 m, but can be up to 30-m-thick. The dikes occur as swarms tens of meters wide with a spacing of 300 to 400 meters. Dike geometries are variable and locally very complex. Horsetail, anastomosing and sigmoidal orientations are observed. Dike contacts can be faulted with tens of meters of dextral displacement. The dikes weather to clay-rich, dark brown, crumbly rocks. The diabase dikes range in age from 222 to 220 Ma (Dooley and Smith, 1982).

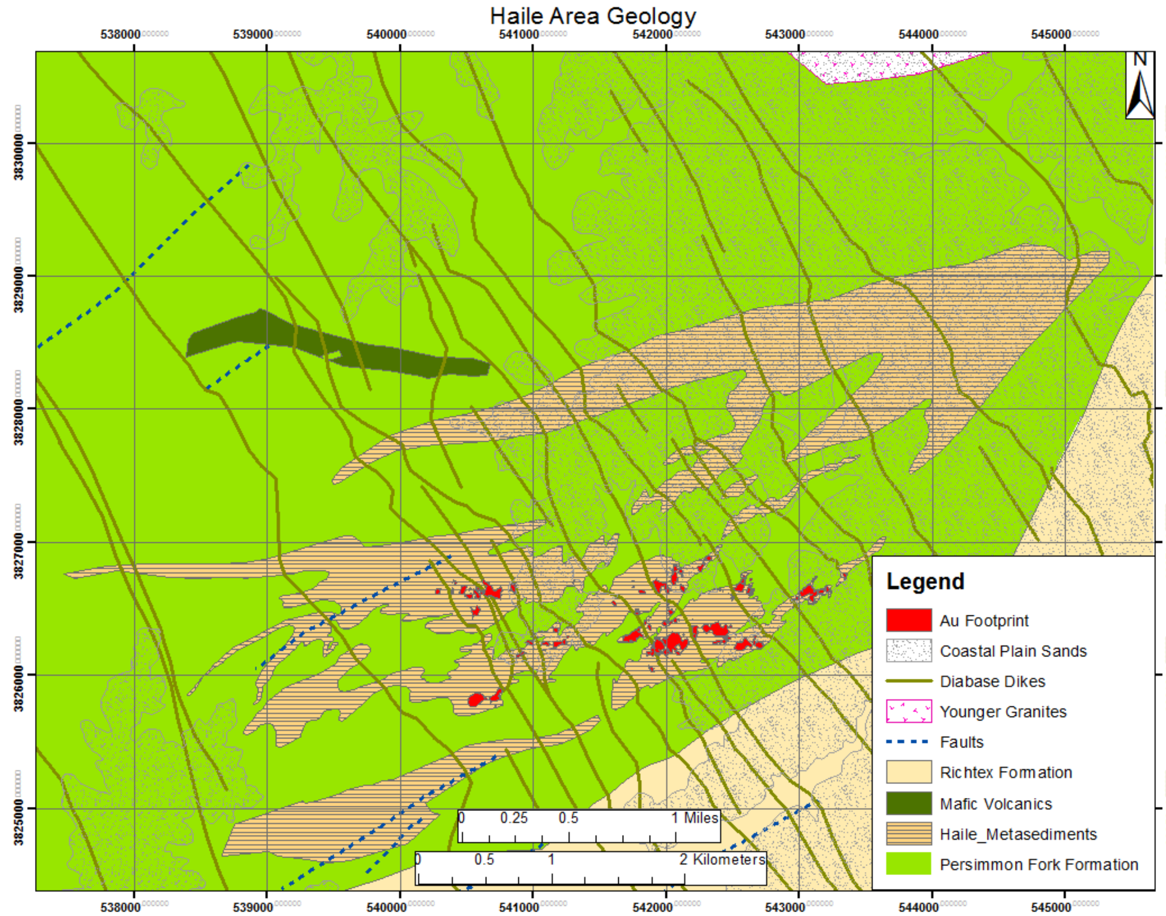
Saprolite

The entire region is covered by saprolite. Saprolite is a thick, structureless, unconsolidated, kaolin-rich, red-orange to white residuum derived from intense bedrock weathering in sub-tropical climates. Saprolite blanket thickness ranges from 10 to 40 m and is thickest in metavolcanic rocks and along faults. The base of saprolite is irregular and grades downward into partially weathered bedrock.

Coastal Plain Sand

The Cretaceous Middendorf Formation (Nystrom et al., 1991) is a southeastward-thickening apron of unconsolidated sand. Its northwest limit conceals much of the Haile property. The sand is up to 23 m thick at Haile and hundreds of meters thick south of Haile. The basal portion contains 10 to 60 cm thick layers of red-brown ferricrete and quartz pebbles in a sandy matrix. The middle unit is white to red sand with a kaolinite matrix. The upper unit is a clean tan to white quartz sand.

Figure 7-3 shows a geologic map of the Haile area with gold zones.



Source: OceanaGold

Figure 7-3: Geologic map of the Haile area with gold zones (UTM NAD83 Z17N)

7.2.2 Structure

The structural history of Haile is complex and long-lived and is characterized by strong ductile deformation. Penetrative strain overprints the Richtex and Persimmon Fork rocks with strong foliation, slaty cleavage, open to isoclinal folding, and local shearing. The foliation surface results from alignment of mica minerals. The more massive portions of the Persimmon Fork are less foliated but micas within them are aligned. Foliation generally strikes northeast and dips moderately northwest. Foliation intensity increases along the sedimentary-volcanic contact, and is strongest in the laminated siltstones where gold mineralization is best developed. Gold-pyrite-silica-rich fluids were preferentially precipitated in more permeable and structurally deformed metasediments.

The Haile gold deposits are exposed along the axis and limbs of a 5 km long by 1.5 km wide ENE-trending asymmetric anticlinorium. The flatter northwest limb of the anticlinorium dips 30 to 50 degrees northwest and includes the Champion and Mustang deposits and portions of the Ledbetter deposit. The steeper southeast limb of the Haile anticlinorium dips 60 to 80 degrees southeast and contains the Palomino, Red Hill, Snake and Horseshoe deposits. The anticline core consists of strongly folded and foliated metasediments that host the largest gold deposit at Ledbetter, and also is mineralized at the 601, Small, Mill Zone, and Chase Hill deposits. Axial planar shearing is manifested as ENE-striking, 40-60^{0}NW-dipping faults that focus high-grade zones > 3 g/t Au. Fold axes generally strike N40^{0}E to N70^{0}E and fold wavelengths range from meters to hundreds of meters. Fold axes typically have shallow to moderate ENE plunges.

Brittle deformation is less common at Haile and includes shear zones with pressure shadows, passive-slip planes, ribbon quartz along slip planes, mica fish, and anastomosing foliation surfaces. Small-scale, brittle offsets have been mapped with both ENE and NW trends. Brittle deformation is most intensive along NW-trending swarms of diabase dikes.

Stratigraphic offset along NW-striking faults down steps stratigraphy from west to east across the Haile district. The metavolcanic-metasediment contact at Champion occurs at the surface in the western portion of the district and is over 500 m deep 4 km to the east near the Horseshoe deposit. Depth and position of the contact is complicated by folding and hence stratigraphic displacement by faulting is difficult to quantify.

Quartz (+_calcite) veins are locally common at Haile, notably at the Mill Zone, Haile and Horseshoe deposits. Veins are rarely mineralized. Veins typically strike N-S to NNE and dip steeply east. Crustiform textures indicative of boiling are rarely observed. Two phases of quartz veins are recognized at Haile, an early diffuse saccharoidal vein phase is deformed, and finely crystalline with white to very light gray colors caused by finely disseminated pyrite. Rarely, the pyrite is interlocked with visible gold up to 0.5 mm in size, e.g. Horseshoe. The distribution of these early veins is poorly understood. The more common quartz (+ calcite) vein type consists of massive, cross-cutting white bull quartz veins that are not mineralized. Prominent NNE-striking quartz ribs 0.3 to 2 m in width are observed in the Sericite pit and in the south wall of the Mill Zone pit.

7.2.3 Mineralization and Alteration

Haile gold mineralization occurs as an en echelon 5 km long by 1.5 km wide cluster of moderately to steeply-dipping ore lenses within a ENE-trending anticlinorium. Eleven named gold deposits are recognized at Haile. From west to east these include Champion, 601, Small, Mill Zone, Haile, Ledbetter, Mustang, Red Hill, Palomino, Snake and Horseshoe. Ore body geometry, depth, size, grade, mineralogy and alteration are variable between deposits. Ore zone geometry is strongly controlled by post-mineral folding and position within the Haile anticlinorium. Some of the deposits coalesce, especially in the central part of the district around the large Ledbetter deposit. Ore lenses are typically 50 to 300 m long, 20 to 100 m wide, and 5 to 30 m thick.

Gold mineralization at Haile is mostly hosted by folded laminated siltstone and greywacke of the upper Persimmon Fork Formation and is capped by less permeable volcanic rocks. Mineralization is typically within 100 meters of the sediment-volcanic contact. Mineralized zones at Ledbetter, Red Hill and Snake are partly hosted in volcanic rocks.

Gold mineralization at Haile is disseminated and occurs in silicified and pyrite--rich metasediments with local K feldspar and molybdenite. Mineral zonation is a quartz-sericite-pyrite+-K feldspar+-gold (QSP) → , sericite +- pyrrhotite → propylitic (chlorite-calcite-epidote) haloes. QSP mineralized zones are tens of meters wide. Sericite envelopes range in thickness from tens to hundreds of meters and are controlled by protolith, structural permeability and post-mineral folding. Within the mineralized zones, quartz is dominant (60% to 80%), pyrite is moderate (1% to10%), and sericite is variable at 5% to 20%. Two silicification events are observed in the mineralized zones. Early massive silicification is finely disseminated to diffuse. Later silicification is manifested as matrix fill in tectonic and hydrothermal breccias and as stock work veinlets. Sericite alteration is commonly expressed as sericite schists due to sericite replacement of micaceous layers in metasediments, imparting a tannish white color. Bleaching and/or argillisation is weakly developed within and adjacent to sericite zones. Propylitic alteration is characterized by increased chlorite (5% to 20%) and a mottled texture with blebs of 3 to 5 mm calcite aggregates. Late calcite +- quartz veining is focused along fault zones.

High-grade zones >3 g/t Au are characterized by intense silicification, anastomosing quartz veins, hydrothermal breccias and >1% fine-grained pyrite. Pyrite grain size is typically <20 microns and occurs as stringers, lenses and banded layers, including graded beds and reworked sulfide sediment. High grade zones are focused where ENE faults coincide with anticline axes in folded metasediments adjacent to the overlying metavolcanic rocks. The Horseshoe deposit averages over 4 g/t and occurs within a tight anticline dissected by ENE-trending, NW-dipping faults.

Oxidation at Haile extends to depths of 20 to 60 meters and is deepest along faults and in folded volcanic rocks. Hematite and goethite are strongest near surface, accompanied by saprolite, and decreased at depth to weak joint stains.

Gold spatially correlates with molybdenite, silver, arsenic, antimony, molybdenum, and tellurium at Haile (Mobley et al., 2014). Arsenopyrite, chalcopyrite, galena, and sphalerite are rarely associated with gold mineralization.

Detailed ore microscopy and scanning electron microscopy show that the gold occurs as native gold, electrum, within gold-bearing tellurides and intergrown with fine-grained pyrite (Foley et al., 2001). Pyrite-pyrrhotite intergrowths are observed and can occur in quartz veins. The pyrite is usually present as finely disseminated euhedral to subhedral grains or aggregates. Coarser cubic pyrite >0.5 mm is not associated with gold mineralization except where it overprints fine-grained disseminated pyrite. Molybdenite occurs primarily on foliation surfaces and as dispersed fine-grained aggregates in silicified zones. The Haile molybdenite has been dated by Re-Os isotopes at 553.8 +- 9 Ma and 586.6 +- 3.6 Ma (Stein et al.., 1997). The first Re-Os age closely approximates the zircon crystallization age of 553 +- 2 Ma reported by Ayuso et al. (2005), indicating that molybdenite mineralization was concurrent with Persimmon Fork volcanism. Seven recent Re-Os molybdenite ages from Haile (Mobley et al., 2014) yield ages ranging from 529 to 564 Ma. Four of these samples give a weighted age of 548.7 + 2 Ma, and hence slightly post dates 555 to 551 Ma volcanism.

8 Deposit Type

Hundreds of gold deposits are located along a 600 km long northeasterly trend that extends from Georgia to Virginia (McCauley and Butler, 1966, Butler and Secor, 1991). Most of these deposits are small prospects worked or explored along narrow quartz veins. The larger gold deposits are located at or near the contact between volcanic and sedimentary rocks, including the Haile, Brewer, Barite Hill and Ridgeway mines. Brewer is unique in the region and is classified as a high sulphidation gold system with volcanic and breccia-hosted gold accompanied by quartz, pyrite, topaz, enargite and chalcopyrite. Gold mineralization at Barite Hill contains the assemblage of pyrite-chalcopyrite-galena-sphalerite and is characteristic of a submarine, high-sulphidation volcanogenic massive sulfide deposit. Haile and Ridgeway are similar whereby gold mineralization is hosted by silicified siltstones in sheared and foliated Neoproterozoic rocks. Ridgeway mineralization was multi-phase, with cleavage-parallel auriferous quartz-carbonate-sulfide veins formed during early heterogeneous ductile deformation, associated with often intense feldspar-destructive quartz-sericite and propylitic alteration (Moye, 2012, Gillon, 2012).

8.1 Haile Genetic Model

Numerous genetic models have been proposed for the Haile gold deposit. The concept of a genetic link between hydrothermal alteration and the formation of gold deposits due to volcanic arc magmatism was first applied to Carolina terrane gold deposits in the 1970. Worthington and Kiff (1970) concluded that a genetic link must exist between ore genesis and volcanism in the Carolina terrane due to their volcanic host rocks. Spence et al. (1980) found a genetic link between gold mineralization hosted within siliceous and pyritic zones and intense alumina alteration which produced kaolinite and sericite-rich zones stratigraphically above mineralized zones and interpreted these as analogous to features observed in modern hot springs. Hardy (1989) concluded that fluids deposited silica, K-feldspar, pyrite, and gold in breccia zones, while depositing silica and gold in the adjacent host rock. Speer and Maddry (1993) interpreted intense alumina alteration noted by Spence et al. (1980) to be the effects of saprolitic weathering in warm, humid climates. Feiss et al. (1993) built on the model of Spence et al. (1980) by proposing that hot spring-type mineralization must have occurred under extension in a back-arc setting based on oxygen isotope data, which they interpreted to mark a shift from a subaerial to submarine environment. Maddry and Speer (1993) proposed an exhalative model for mineralization at Haile whereby gold deposition resulted from hydrothermal fluids venting to the seafloor to produce stratabound ore bodies in marine volcaniclastic rocks.

Several workers concluded that syn-tectonic gold mineralization at Haile was structurally controlled and formed during Neoproterozoic deformation. Tomkinson (1988) proposed that Haile was an orogenic deposit based on textural and structural connections between gold mineralization and shear zones. Hayward (1992) emphasized the importance of folds in controlling the location of ore formation. in anticlinal fold hinges. Hayward (1992) also noted that alteration zones at Haile are generally discordant to bedding and commonly display symmetrical patterns around ore bodies, Hardy (1989) and Worthington (1993) interpreted the features observed by Hayward (1992) and Tomkinson (1988) as evidence for the remobilization of pre-existing mineralized horizons, causing gold enrichment along structurally controlled pathways during deformation which postdated the primary phase of mineralization at Haile. Gillon et al. (1995) proposed a model that invoked early gold mineralization and remobilization during deformation. O'Brien et al. (1998) proposed that the gold deposits were formed during arc-related volcanic activity in a hydrothermal system. Foley et al. (2001) observed multiple generations of pyrite in Haile ores and concluded that disseminated pyrite and gold mineralization were contemporaneous with host volcanic rocks and volcaniclastic sediments.

Gold mineralization (~549 Ma) at Haile slightly postdates volcanism, which precludes syngenetic and volcanogenic models, and predates deformation. Haile ore zones are strongly folded and locally faulted. Pressure shadows around pyrite grains, folded mineralized zones, stretched pyrite and pyrrhotite grains and flattened hydrothermal breccia clasts indicate that the mineralization is pre-tectonic. Mineralized zones were subsequently foliated, folded, sheared, faulted, and dissected by diabase dikes. Similar timing for gold mineralization and peak magmatism in the Haile and Ridgeway areas indicates that the hydrothermal systems that produced these deposits were driven by magmatism and were not the product of collision, orogeny, and/or a related metamorphic event (Mobley et al., 2014).

8.2 Haile Geological Model

The Haile geological model consists of 3D wireframes created in Vulcan for the following five units, from youngest to oldest: Coastal Plain Sands, saprolite, diabase dikes, metasedimentary rocks, metavolcanic rocks. Base of oxidation and silicification were also modeled. Further modeling is planned to improve the model, notably for faults and lithologies. Most modelled dikes are complex with anastomosing geometries and splays. There is local evidence of cross-cutting dike relationships at Horseshoe as influenced by the pre-existing NW-striking fault system where a cymoid loop (right-lateral, mostly strike-slip) geometry is interpreted.

9 Exploration

9.1 Pre-Romarco

Modern exploration, development, and mining activity on the Haile property began with mapping in 1970 (Worthington and Kiff, 1970). Between 1973 and 1977, Cyprus conducted an extensive exploration program consisting of surface geophysical surveys, trenching, geologic mapping, auger drilling, core drilling, air-track drilling, and metallurgical testing. Cyprus calculated the Haile resources at 186,000 ounces (5,785 kg) of gold with an average grade of 2.13 g/t. Resources reported in this section do not conform to the standards of NI 43-101 and are included only as part of the historic record.

Between 1981 and 1985, Piedmont explored the historic Haile Mine and surrounding properties with core and reverse circulation drilling, surface geophysics, soil sampling, trenching, and rock-chip sampling. Piedmont's total drilling footage was 69,647 meters, much of which was for mine development. Piedmont mined several Haile property deposits from 1985 to 1992, producing about 86,000 ounces (2,675 kg) of gold.

In 1991, Amax performed an extensive exploration program on the Haile property under an exploration option with Piedmont. In 1992, Amax and Piedmont formed HMC as a joint venture, and from 1992 to 1994 HMC (the operating company) completed a program of exploration/development drilling using core and reverse circulation drilling, Mineral Resource estimation, and technical report preparation (Wells and Wolverson, 1993). The Ledbetter deposit was discovered and the Mill and Snake areas were expanded.

Kinross acquired Amax in 1998, assumed Amax's portion of the HMC joint venture, and later purchased Piedmont's interest. Kinross performed no exploration activities on the property and limited their operations to a highly successful reclamation program from 1998 to 2007.

9.2 Romarco

Romarco completed the Haile property acquisition in October 2007. By February 2008 Romarco had confirmed the quality of historical drilling and assay data and turned their effort to exploration and resource expansion drilling. During its ownership, Romarco expanded the resource and reserve of the property by five-fold. This report documents the results of the drill program achieved to date with assay data available through November 17, 2011 and subsequently by OceanaGold, as described below.

9.3 OceanaGold Exploration

OceanaGold purchased the Haile property from Romarco in October 2015 and continued the drilling programs designed to expand resources and reserves at Haile. Mine development drilling is ongoing.

9.3.1 Geologic Mapping and Surface Sampling

Numerous workers have performed geologic mapping and surface sampling in and around the Haile area. Mapping is challenged by poor bedrock exposure due to extensive saprolitic weathering, Coastal Plain Sand cover, and dense vegetation. Outcrop is estimated at only 1% to 2% in the Haile area. Most of the better-quality mapping was focused within the excavations related to mining. Bell completed a regional geologic map for the Kershaw quadrangle in 1980. More detailed mapping was conducted at

Haile by Spence, Kiff, and Maye, who constructed a detailed geologic map for the mine site in 1975. Subsequent detailed geologic mapping was done by Taylor in 1985 and Cochrane in 1986. Masters' degree dissertations by Tomkinson in 1985 and by Hayward in 1988 included extensive geologic mapping. Geologic mapping at the Mill Zone pit resumed with mining in 2016 by OceanaGold geologists.

Historical mapping data has been scanned and loaded into the Vulcan software for structural interpretation, exploration planning, and geologic modeling. The use of the structural dataset in conjunction with the drilling dataset has provided the foundation for a 3D computerized geologic model. This model continues to be used successfully to expand the resources and reserves at the Haile property. Surface samples have been compiled into an Access database and evaluated by OceanaGold. Over 5,000 samples have been compiled based on location, sample type (rock chip, saprolite, soil, stream sediment), rock type, alteration and assay. QA/QC data are generally lacking for these samples and most were assayed only for gold.

9.3.2 Geophysics

Numerous geophysical surveys have been conducted at Haile since the 1970s. The following geophysical methods have been applied at Haile:

1. Gravity – maps density and therefore lithology
2. Airborne magnetic – maps diabase dikes, mafic dikes, granite, faults
3. 3D inversion magnetic model
4. Airborne EM – maps conductor surface features like power lines, granite and silicification
5. Induced polarization (IP)/resistivity – maps clay, sulfide and silicification

Numerous IP/resistivity surveys have been conducted at Haile, including surveys by Piedmont in 1975 and 1989, 2015 Romarco surveys at Champion, Mill Zone, Ledbetter, and Horseshoe, and 2016 OceanaGold surveys at areas adjacent to Haile. Geophysical surveys conducted by Piedmont in the late 1980s include ground magnetics and dipole-dipole IP/resistivity methods that led to discovery of the Snake deposit (Larson and Worthington, 1989). The ground magnetic data were acquired in a patchwork fashion and were not corrected for diurnal changes. Weak IP responses to 10 milliseconds can map sulfides at depth and large silicified bodies. The dipole-dipole IP/resistivity data were reprocessed by OceanaGold in 2016 and used for drill targeting and geologic modeling (Weis, 2016).

Regional gravity survey and aeromagnetic data have been downloaded from the South Carolina data repository (Daniels, 2005). These were supplemented by more detailed gravity stations in 2009 and 2010 by Romarco along roads in the Haile area and as transects over Haile deposits.

Airborne EM and magnetic surveys were flown by Aeroquest for Romarco in 2010 over the Haile-Brewer area on 50 m and 100 m spaced flight lines with a bearing of 150 to 330 degrees. The magnetic data are capable of mapping the diabase dikes and granite plutons but do not differentiate the older units. Proprietary 3D inversion modeling was conducted by OceanaGold in 2016 to depths of 1,500 m using airborne magnetic and EM data.

10 Drilling

10.1 Type and Extent

Drilling at Haile commenced in the 1970s and has continued intermittently to the present by several companies. The database used for this resource estimate was extracted from the acQuire database on July 29, 2016, which includes 4,301 drill holes totaling 598,276 m of drilling. Not all the drilling was used for estimation of the block model because it included shallow auger holes. RC and core drilling by Romarco was continuous from 2008 to 2012 and then resumed in 2015 after a hiatus due to permitting and lower gold prices. Drilling at Haile has continued since early 2015, almost entirely as core drilling.

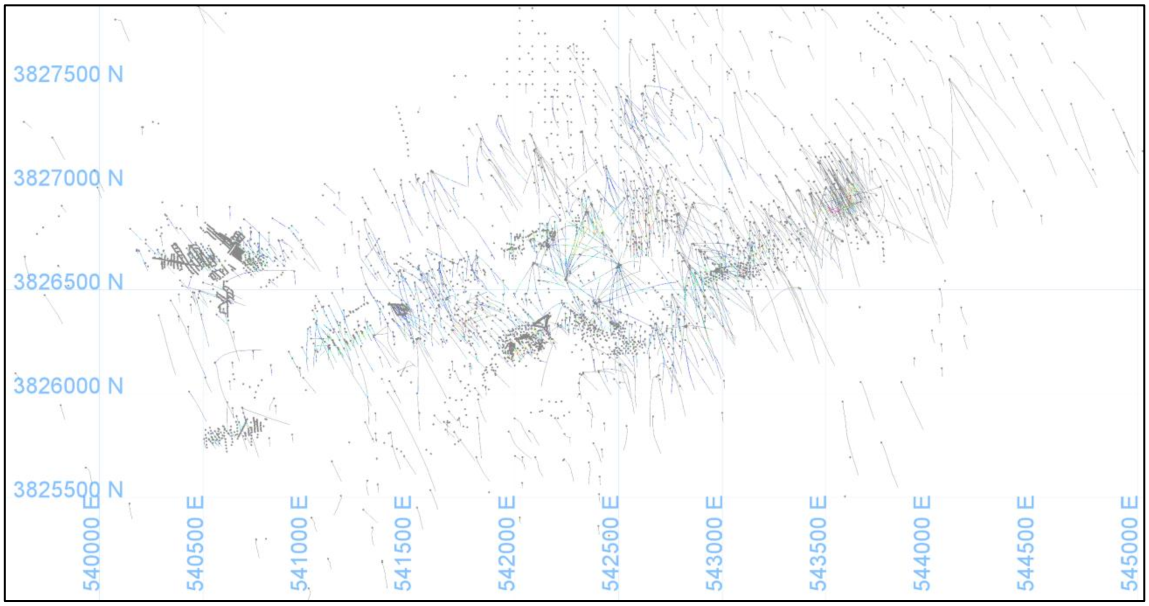
As of November 19, 2015, Romarco had drilled 502,199 m (1,647,637 ft) on the property. From December 2015 to September 2016 a 65-hole infill drilling program for the Horseshoe deposit was completed by OceanaGold Exploration. Prior to Romarco, 113,044 m (370,879 ft) of fire assayed drilling was completed by previous property holders including Cyprus, Gold Fields Mining Corp, Piedmont, Westmont Mining, and the Piedmont/ Amax HMC joint venture. A portion of the early drilling has been mined out and has little impact on the remaining resource estimation. Some of the Piedmont and Cyprus drill holes were assayed by cyanide soluble methods to determine cyanide amenability. That information has not been used in the determination of resources and only those intervals with fire assay from those previous property holders have been used.

IMC completed a comparison of historic drilling to Romarco drilling and found that the old and new data can be commingled if it has been fire assayed. Within the fire assayed data, 28% of the holes are core and 72% are RC. A few fire assays (301) are from air track and doodlebug drilling. They amount to 0.2% of the database and are not a significant sample set.

Drilling at Haile since RC hole number 1502 and all DDH holes since hole number 289 have received down hole surveys, which amounts to 32% of the RC holes, 100% of the core-tail holes, and 89% of the diamond drill holes. Since all of the surveyed drill holes deviate clockwise, an algorithm was developed to adjust the down-hole survey of the historical drill holes to reflect their likely deviation.

RC holes by Romarco have a diameter of 159 mm using a company owned Schramm 685 drill using 500 psi at 1350 cfm. RC drilling was typically used to drill down to mineralized zones, and then NQ core holes were drilled. RC drilling was also used for sterilization drilling for waste dumps, the process plant and the tails storage area. Drilling since 2015 has exclusively used HQ core drilling through saprolite and weathered zones, and then NQ core drilling in fresh bedrock for both waste and ore zones. Core drilling by Romarco and OceanaGold has been performed using company owned LF90 drills. Core recovery in fresh rock is typically 95% to 100%. Core recovery in saprolite ranges from 10% to 50%. Minimal ore has been identified in saprolite.

Figure 10-1 shows a drill hole location map as of January 2017.



Source: OceanaGold

Figure 10-1: Haile drill hole location map as of January 2017

10.2 Sample Collection

Romarco and OceanaGold used both reverse circulation (RC) and diamond drilling (DDH) methods at Haile. This section describes the sampling procedures applied to both drill methods. The sample procedures applied to the historic drilling at Haile are not well known and have not been documented. IMC completed a statistical comparison between the historic information and the recent drilling to provide verification of the reliability of the historic drilling.

John Marek, the qualified person for IMC on behalf of Romarco, reviewed the sample preparation, analysis, and security utilized at Haile and found the procedures to be proper for determination of Mineral Reserves and Mineral Resources. The results of quality control sampling reported below are summarized in Section 12.

Both RC and DDH have been used for the resource estimates at Haile. This section will describe the sampling procedures applied to both data collection techniques. Historical drilling prior to 2007 accounts for approximately 30% of the data.

Romarco has been drilling at Haile since 2007. The techniques described in this section reflect the procedures applied by Romarco and OceanaGold during the period up to 2017.

Bruce van Brunt, Technical Services Manager for HGM, and John Jory, Director of Exploration for OceanaGold, are the competent persons for this section. They have reviewed the sample preparation, analysis, and security utilized by HGM and find the procedures to be proper for determination of Mineral Resources and Ore Reserves.

Reverse Circulation (RC) Drilling

The reverse circulation drilling at Haile typically uses 6.25-inch (159 mm) drill bits. The RC rigs are equipped with a cyclone and a rotary splitter. Most RC drilling at Haile is in wet conditions. Water injection is typically 0.25 to 0.31 ls-1 (4 to 5 gpm) above the water table and decreases to 0.06 ls-1 (1 gpm) when groundwater is encountered.

Sample sizes are between 3 to 7 kg (20 and 30 lbs) with a minimum requirement of 3 kg (15 lbs). The standard size reflects a 15 to 20% split of the total drilled volume. Drill intervals are generally 1.5 m (5 ft) intervals.

The following paragraphs describe sample procedures as reported by Romarco personnel. IMC observations during the site visit confirmed the application of these techniques.

For each 1.5 m (5 ft) interval, a sample container is placed on top of the splitter table to catch the flow from the sample splitter. Labeled, sample bags measuring 500 mm by 600 mm (20 inch by 24 inch) are placed in 20 L (five to seven-gallon) plastic buckets. Multiple 6 mm (quarter-inch) holes are predrilled in the plastic buckets to reduce the suction of a full sample bag and allow limited water drainage. The top of the sample bag is folded securely over the edge of the bucket. This is the sample container that is placed under the splitter to catch the sample discharge. Flocculant is added to each sample bag as it is placed on the splitter table to aid in precipitating fine material from the sample. As one sample container fills, another sample bag is prepared in advance and staged near the splitter table. On the driller's signal, the sample containers are switched instantaneously at the break between 1.5 m (5 ft) drill intervals.

Sampling during advancement of each twenty-foot (6.1 m) rod is a continuous process. Sample timing is metered by the count of the driller, as determined by drill speed and sample return rate. After each rod break, a new rod is attached and the borehole is thoroughly flushed. The driller should raise the bit slightly off bottom and blow the borehole clean before beginning the next interval. Once the sample return is clean, the bit is lowered and drilling begins on the next twenty-foot rod. Then, the driller counts the time it takes for the discharge water to turn from clear to muddy, which approximates the return rate of samples to the surface. Markings on the drilling rig feeder cable denote five foot intervals. When the feeder cable indicates the completion of the 1.5 m (5 ft) sample interval, the driller counts the measured return rate to allow the last sample material to reach the surface.

The rod break depth is determined by the drilling rig set-up and may vary with every drill hole. The rod break generally occurs within a 1.5 m (5 ft) sample interval. The sample collected over a rod break should be removed from beneath the sample splitter during borehole flushing. Following the addition of a new rod and subsequent flushing, the sample container is replaced and drilling continues. During the rod break, the sampler should clean the splitter, check the splitter plates, measure the pH and temperature of discharge water, and keep current with logging. For rod breaks occurring at shift changes, the crew is mindful of the incomplete sample and communicates its location to the next crew. Rod additions, timing, and bit changes are recorded in the drilling progress log. Filled sample bags are typically kept at the drilling rig during each shift. The samples can be stored on the ground or in the bed of a pickup truck to begin water drainage. At the end of each shift, the samples are transported to the sample storage area for initial drying.

During each drilling interval, a metal mesh-screened strainer (rice/pasta strainer) is placed on the splitter table beneath the waste stream to obtain a representative chip sample for geologic logging. The lithologic sample is collected from the waste discharge material to avoid biasing the assay sample partition. A small portion of the sample is placed in, plastic chip trays for geologic logging. Chip trays are labeled with the drill hole number and 5-ft (1.5 m) depth intervals by a permanent marker. Each chip tray has twenty compartments and therefore represents 100 ft (31 m) of drilling. Chip tray compartments typically measure about 3 cm x 2 cm by 2 cm deep.

Sample bags are collected at the end of each shift and transferred to the Haile sample storage area for initial drying.

Diamond Drilling

Diamond core drilling is by wireline methods and generally utilizes HQ and NQ size core with 63.5 mm and 48.3 mm diameters respectively. Drill rods are ten feet (3.1 m) long. Core is transferred from the core barrels into plastic core boxes at the drill rig by the driller.

Each core box can hold up to ten feet (3.1 m) of core stored in five rows each two feet (0.6 m) long. Core is gently broken by hammer as required to completely fill the boxes. Hole numbers and drill depths are marked on the outside of the core boxes and interval marker blocks are labeled and placed in the core box. Boxed whole core is covered with plastic lids and is transported to the core shed for logging and sampling by OceanaGold personnel.

10.3 Collar Locations and Downhole Surveys

Drill hole numbers are assigned and maintained by company geologists via a Microsoft Excel (Excel) tracking spreadsheet that records location, depth, azimuth, dip, start and end dates. Historical drill hole collar surveys by Piedmont and Cyprus were surveyed by theodolite and recorded on paper. Haile drill hole collars since 2007 have been surveyed by company surveyors using digital GPS methods. Survey equipment include a SPS985 Rover receiver and a TSC3 SCS900 data receiver. Surveys utilize an on-site base station and have sub cm accuracy. All surveys are performed by qualified OceanaGold personnel.

Collar surveys are downloaded by the surveyors as .csv files and forwarded to the geologists. Collar coordinates are verified against planned coordinates by the geologist overseeing the drilling, and then imported into the acQuire database. Drill hole collars are marked with 10 cm PVC pipe and a metal tag. After verification of collar coordinates, drill sites are rehabilitated within 60 days as mandated by the operating permit.

Historical drill holes prior to Romarco in 2007 were not surveyed for downhole deviation. Since 2007, all angle holes have been surveyed by Romarco and now OceanaGold technicians using the Reflex EZ Trac survey tool. The tool is stored in a case at the Haile OceanaGold office and is calibrated at the Reflex office in Tucson, Arizona once a year. Multi-shot surveys are recorded for azimuth and dip every 6.1 m (20 ft). Survey data are downloaded and verified using Reflex Sprocess software by geologists overseeing the drilling. Magnetic intensity values are also reviewed to explain spurious data. Upon verification, the data are imported into the acQuire database.

Original paper or printed survey records are located in the OceanaGold exploration office and are stored digitally in the acQuire database. As part of a company-wide metrification process, OceanaGold transformed all surface and drill hole data from the South Carolina NAD27 coordinate system to the UTM NAD83 zone 17N system in November 2016. Coordinate transfer was verified by geologists and engineers and no issues were identified. Coordinate transformation was supported and utilized by SRK personnel during the Horseshoe underground evaluation.

10.4 Significant Results and Interpretation

The database used for this technical report includes 4,301 holes in the Haile district which are securely stored in OceanaGold's acQuire database. Drill hole collar locations, downhole surveys, geological logs, geotechnical logs and assays have been verified and used to build 3D geological models and in grade interpolations. Geologic interpretation is based on structure, lithology and alteration as logged in the drill holes. The disseminated style of gold mineralization at Haile enables relatively robust geologic models to be produced, despite complexities by folding.

Haile has a large percentage (28% of drill meters) of historical holes drilled before 2007 for which sampling methods have not been documented. Collar coordinates were written on paper and therefore some field note errors may have occurred during surveys or copying into databases. Historical holes were also not surveyed for downhole deviation and their location in 3D space cannot be verified. QA/QC records are sparse for the historical drill holes and QA/QC methodology was not documented. Numerous labs were utilized, including ACME,

Significant mineralization has been recorded in drill holes at eleven named gold deposits at Haile within a 3.5 km by 1 km area since drilling commenced in the 1970s. Drill hole spacing typically ranges from 30 to 60 meters, and drill gaps are wider between mineralized zones. Resource drilling at Haile has predominantly been conducted by core holes angled to the southeast at dips of -50º to -65º perpendicular to regional foliation. Intersection angles between drill holes and mineralization are typically in the range of 60° to 80°. Significant drill hole intercepts therefore range from 70% to 100% of true mineralized widths.

Hole depths typically range from 120 to 500 meters and are dependent on ore depth. Sample interval lengths are typically 1.5 meters (5 feet) but can vary based on geological logging. True thickness of mineralized intervals is dependent on the intersection angle by drill holes with the foliation and shear zones; the intersection angle ranges from 60 to 90 degrees in holes angled to the southeast. Some holes have been angled to the north, northwest and west to infill drill gaps where collar access is restricted by infrastructure (leach pads, pits, haul roads, lakes, wetlands). High grade zones >3 g/t Au occur in strongly silicified and pyritic rocks and are enclosed by lower grade haloes with weaker or absent silicification and 0.5% pyrite. The only exception is where diabase dikes cut mineralized zones, which create very sharp ore-waste boundaries.

11 Sample Preparation, Analysis and Security

11.1 Sample Preparation for Analysis

11.1.1 On-site sample preparation

RC Samples

The reverse circulation (RC) sample bags from the truck are transferred to the Haile sample handling facility where they are prepared for shipment to a lab. RC samples are prepared at either the Kershaw Mineral Lab (KML) in Kershaw, South Carolina or the AHK Geochem (AHK) preparation facility in Spartanburg, South Carolina.

Samples follow one of two paths:

1. Some samples are weighed and sample number tags added to the bags. The samples are poured through a Jones splitter to reduce the size to roughly 2.7 kg (6 lbs) for shipment to the sample lab. Coarse rejects are kept in their original sample bags and stored on site on pallets.
2. Alternatively, samples are staged at Haile and placed in containers for direct shipment to KML or AHK.

Core Samples

At the core logging facility, the core is cleaned, measured, and photographed. Geotechnical and geologic logging are completed on the whole core. All logging and sampling handling is conducted by OceanaGold personnel. Data collecting during core logging include structure, rock type, alteration, mineralogy, Rock Quality Designation (RQD), core recovery, hardness and joint condition. Alteration is logged as relative intensity and includes weak, moderate and strong categories. Mineralogy is visually estimated to the nearest 0.1%. Standardized templates are used for all logging with drop down menus. Geologists routinely review core together and compare notes to ensure accuracy and consistency. Density samples are collected every 10 m (33 ft) and use the water immersion method to measure specific gravity. Competent core at Haile does not require plastic or wax coatings for density measurements. Paper logs are entered into an Excel spreadsheet and then imported in the acQuire database by the admin assistant. Logs are periodically checked by the geologists for accuracy and completeness. Tablet-based geology logging in Excel was initiated in 2017 and enables logs to be directly uploaded into acQuire.

Logging is conducted in the Imperial system using feet due to the 10-ft drill rods. Data are converted to metric units after being imported into the acQuire database. The logging geologist assigns the sample intervals and sample numbers prior to core sawing. Sample ID tags are placed in the core boxes. Sample lengths are typically 5 feet (1.5 m) and can range in length from 1 ft (0.3 m) to 10 ft (3.1 m). Geologic sample breaks may be selected by the geologists based on contacts or structural boundaries. 'No sample' intervals are marked by orange flagging tape in surficial fill or rubble zones and in massive barren diabase dikes exceeding 50 feet (15 m) in thickness.

Core is sawed in half along the core axis using circular masonry blades and then placed into sample bags labelled with the sample ID. Paper ID tags are also placed into the bags. Saprolite zones are manually cut with a putty knife. The saw or knife are cleaned between each sample. A brick or barren rock sample is sawed between intervals to minimize cross-contamination. The cooling water for the saw is not recycled and is discharged into a permitted pond.

Core samples are delivered to the sample preparation facilities. Core is prepared primarily at the company-owned Kershaw Mineral Lab (KML) facility in Kershaw, South Carolina, and secondarily at the AHK preparation facility in Spartanburg, South Carolina.

11.1.2 Off-site sample preparation

AHK, Spartanburg, South Carolina (ISO/IEC 17025 accredited)

Once the samples arrive at AHK in Spartanburg, South Carolina the following procedures were applied:

1. Sample Preparation
2. Inventory and log samples into the laboratory LIMS tracking system
3. Print worksheets and envelope labels
4. Dry samples at 65°C (150°F)
5. Jaw crush samples to 80% passing 2 mm
6. Clean the crusher between samples with barren rock and compressed air
7. Split sample with a riffle splitter to prepare the sample for pulverizing
8. Pulverize a 250 g sample to 90% passing 150-mesh (0.106 mm)
9. Clean the pulverizer between samples with sand and compressed air
10. Ship about 125 g of sample pulp for assay
11. Coarse rejects are returned to Haile for storage
12. The 125 g reserve pulps are stored at the AHK facility in Spartanburg with a seal. They represent an independent chain of custody sample library.

Sample pulps were shipped to the AHK Laboratory in Fairbanks, Alaska for analysis.

Kershaw Mineral Laboratory (KML), Kershaw, South Carolina (ISO/IEC 17025 accredited)

Once the samples arrived at KML, the following procedures are applied:

1. Sample Preparation
2. Inventory and log samples into the laboratory LIMS tracking system
3. Print worksheets and envelope labels
4. Dry samples at 93°F (200°F)
5. Jaw crush samples to 70% passing 10-mesh (2 mm)
6. Clean the crusher between samples with barren rock and compressed air
7. Split sample with a riffle splitter to prepare the sample for pulverizing
8. Pulverize a 450 g sample (± 50 g) to 85% passing 140-mesh (0.106 mm)
9. Clean the pulverizer between samples with sand and compressed air
10. Approximately 225 g of pulp sample is sent for fire assay
11. Coarse rejects and reserve pulps are returned to Haile for storage.

Sample pulps from KML were shipped to the AHK Laboratory in Fairbanks, Alaska for analysis.

11.2 Sample Analysis

The procedures applied at AHK in Fairbanks, Alaska for assay were as follows:

1. Inventory the samples and create worksheets

2. Insert Quality Control samples of two duplicates, one certified standard, and one blank in each batch of 40 samples.
3. Fire assay a 30 g aliquot for gold with 4-acid digestions and Atomic Absorption finish.
4. Analyze 0.50 g samples for Multi-Element by inductively coupled plasma mass spectrometry (ICP-MS).
5. Review the internal QC results and check as required.
6. Review and sign off on final values including the internal check assays.
7. Issue the final report and certificate of assay.
8. Deliver the certificate to the client.

AHK is ISO/IEC 17025 accredited for all facilities that handle Haile samples.

ALS Chemex is ISO 9001 certified and ISO/IEC 17025 accredited. Coarse rejects and returned samples are stored at Haile under the control of company personnel. During off-shift hours, a Deputy Sherriff is on site providing security for the site and sample storage facility.

The procedures currently applied at KML for assay are as follows:

1. Inventory the samples and create worksheets.
2. Insert Quality Control samples of one duplicate, one certified standard, and one blank in each batch of 24 samples.
3. Fire assay 30 g of pulp sample for gold, with Atomic Absorption finish.
4. If the gold assay result from step 3 is greater than or equal to 3 g/t Au, an additional 30 g of pulp sample is fire assayed for gold using gravimetric finish, and 0.5 g of pulp sample is analyzed for silver using a 4-acid digestion with Atomic Absorption finish.
5. Multi-Element ICP analysis is performed as requested.
6. Carbon and Sulfur determinations are performed as requested.
7. Review the internal QC results and perform check assays as required.
8. Review and sign off on final values including the internal check assays.
9. Issue the final report and certificate of assay.
10. Deliver the certificate to the client.

KML is ISO/IEC 17025:2005 accredited for gold and silver assays through the Standards Council of Canada.

The procedures currently applied at KML for assays are as follows:

1. Inventory the samples and create worksheets.
2. Insert Quality Control samples of one duplicate, one certified standard, and one blank in each batch of 24 samples.
3. Fire assay 30 g of pulp sample for gold, with Atomic Absorption finish.
4. If the gold assay result from step 3 is greater than or equal to 3 g/t Au, an additional 30 g of pulp sample is fire assayed for gold using gravimetric finish, and 0.5 g of pulp sample is analyzed for silver using a 4-acid digestion with Atomic Absorption finish.
5. Multi-Element ICP analysis is performed as requested.
6. Carbon and Sulfur determinations are performed as requested.
7. Review the internal QC results and perform check assays as required.
8. Review and sign off on final values including the internal check assays.
9. Issue the final report and certificate of assay.

10. Deliver the certificate to the client.

11.3 Check Assays

Early in the Romarco drill program, samples were sent to the Inspectorate Lab in Reno, Nevada for preparation and assay. Inspectorate is an ISO 9001 certified laboratory. Check assays were sent to ALS-Chemex in Reno, Nevada. Sample analysis procedures at ALS are as follows:

1. Inventory the samples and create worksheets.
2. Insert Quality Control samples of one duplicate, one certified standard, and one blank in each batch of 20 samples.
3. Fire assay 30 g of pulp sample for gold, with Atomic Absorption finish.
4. If the gold assay result from step 3 is greater than or equal to 3 g/t Au, an additional 30 g of pulp sample is fire assayed for gold using gravimetric finish, and 0.5 g of pulp sample is analyzed for silver using a 4-acid digestion with Atomic Absorption finish.
5. Multi-Element ICP analysis is performed as requested.
6. Carbon and Sulfur determinations are performed as requested.
7. Review the internal QC results and perform check assays as required.
8. Review and sign off on final values including the internal check assays.
9. Issue the final report and certificate of assay.
10. Deliver the certificate to the client.

11.4 Quality Assurance/Quality Control Procedures

11.4.1 Standards

Certified standards are routinely inserted at a rate of one in twenty samples (5%) per industry guidelines. Standards used by Romarco and OceanaGold are purchased from and certified by Rocklabs and include six standards of various grades. Five are oxide standards and one is sulphidic.

11.4.2 Blanks

Blanks are routinely inserted at a rate of one in twenty samples (5%). Blanks used by Romarco and OceanaGold include commercially available marble, sand, quartz pebble.

11.4.3 Duplicates

No duplicates samples were collected or analyzed.

11.4.4 Actions and Results

QA/QC data and graphs are generated from the acQuire database. Standards and blanks greater than 20% of the expected value are re-assayed for failed batches (20 samples). Reruns have been acceptable and those values were imported into the acQuire database.

Security Measures

RC coarse rejects and returned samples are stored and secured at Haile where they are under the control of OceanaGold personnel. Pulps are stored in converted turkey barns at Haile with the coarse rejects. RC chip trays are stored in the exploration office. Boxed core is palletized and stored in a sand lot on the south side of the mine property. Pallets are covered by tarps and aluminum tags with hole IDs are attached to each pallet.

11.5 Opinion on Adequacy (Security, Sample Preparation, Analysis)

Sample collection, preparation and analysis are according to industry standards. All labs used by Romarco and OceanaGold are certified to ISO 9001 standard or ISO/IEC 17025 accredited for gold and silver through the Standards Council of Canada. The primary external labs used for check assays at ALS Reno and ALS Tucson are both ISO 9001 certified and ISO/IEC 17025 accredited

Core, pulp and RC sample storage are considered secure based on the opinion of John Jory, Director of Exploration, HGM. Sample transport is by company personnel between secure facilities and by approved couriers to external labs. No significant risks have been recognized for sample contamination or sample exchange.

All Haile drill hole data (assays, logs, surveys) are stored in the secure acQuire database which is managed by the senior database geologist. Assay data can only be imported by the senior database geologist. This geologist has no direct reporting relationships to the Haile geologists. acQuire is an industry certified database that has limited editing capacity beyond the senior database geologist. Data changes are tracked and verified. Strict data importing and verification protocols must be followed to avoid, for example, overlapping or missing intervals, mismatched hole depths in different fields, duplicate hole IDs or sample numbers, and invalid logging codes.

12 Data Verification

This section covers data verification done by various parties.

Section 12.1 and its subsequent parts covers a data verification study done by IMC. IMC did data verification for the drilling from October 2010 thru November 16, 2011 as well as the historic drilling before October 2010.

Section 12.2 and its subsequent parts covers data verification that OceanaGold Exploration did on the drill holes that were used in the underground Horseshoe block model. The report is dated Nov 20th 2016.

Section 12.3 and its subsequent parts covers data verification that OceanaGold Exploration did on the drill holes that were used in the underground Horseshoe block model. Specifically, the subject was "KML and ALS Horseshoe assays comparison". The report is dated November 21, 2016

Section 12.4 and its subsequent parts covers quality control analysis by OceanaGold of Haile data for the date ranges 2011 to 2012 and 2016 to 2017.

- Any other drilling used in the HGM16 model not included in the data sets mentioned above hasn't had data verification applied to it. It would have had the same QA/QC procedures applied to it as the data sets above.

12.1 Romarco Data Verification

The following checks have been applied to the Romarco data by IMC.

- A comparison of certificates of assay from the laboratory versus the Romarco computerized data base to check the reliability of data entry;
- Statistical analysis of the standards samples that were inserted by Romarco for analysis by the assay lab;
- Statistical analysis of the blank samples that were inserted by Romarco for analysis by the assay lab; and
- Statistical analysis of the check samples that were submitted by Romarco to a third-party laboratory

12.1.1 Certificate Check

Certificate checks were completed by IMC in two iterations that correspond to block model updates in October 2010, and November 2011. IMC established a list of drill hole certificates and requested them to be scanned and sent to IMC for a spot check of the data base.

During the October 2010 check, IMC requested the original certificates of assay for 46 drill holes completed by Romarco. The selection of holes was established by IMC to cover the entire life of the Romarco drill program from 2007 through the drilling in the third quarter of 2010. Of the 46-hole selection, 25 were drill holes completed between late 2009 and 2010.

Within the October 2010 data base the 46 holes contained 10,055 assay intervals. Within those intervals, IMC found 11 intervals where the Haile data base did not match the certificate of assay. All 11 discrepancies were in the low grade or trace range. In some cases, they were assigned as no assays in the data base and in others they were assigned as zero values.

IMC obtained certificates of assay for 42 holes that were drilled between the end of 2010 and the close out date for the November 2011 model update. There were 11,046 assay intervals within those holes. There was one interval in drill hole RC1914 where the assay data base did not match the certificate data.

The certificates were missing for 306 intervals out of the total, or about 2.8% of the requested files. Most of the missing intervals were isolated single pages missing out of multiple pages of certificates, implying they were simply skipped.

12.1.2 Statistical Analysis of Romarco Standards

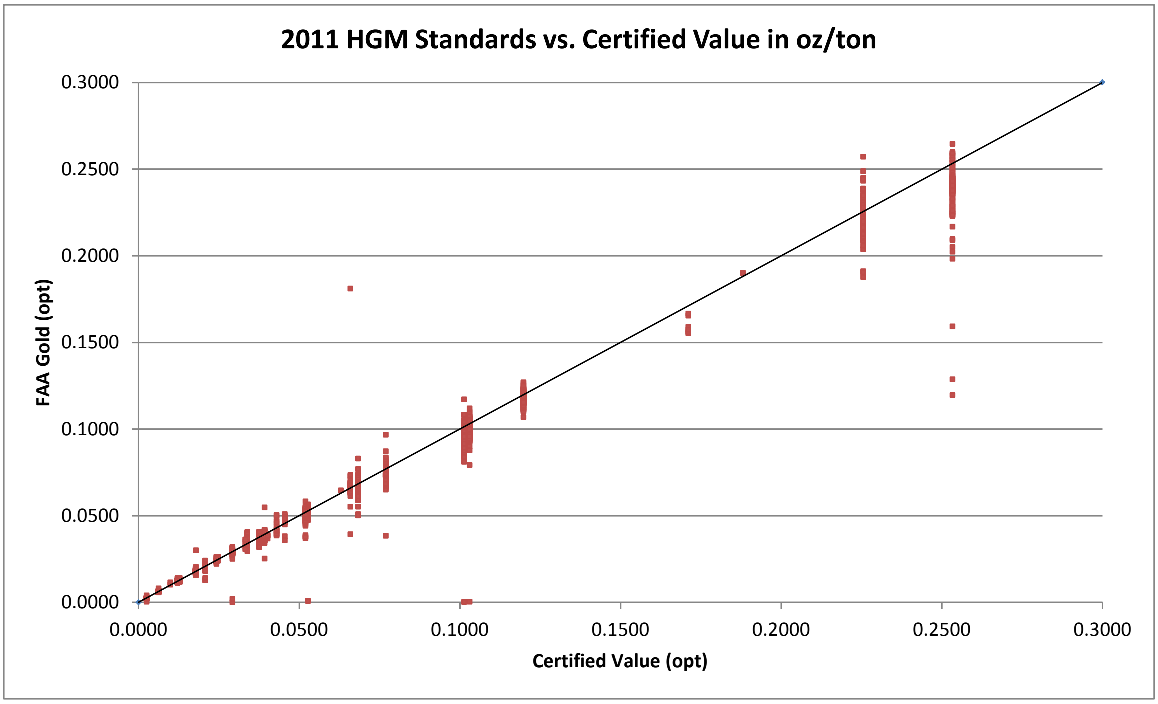
Certified standards are inserted by Haile geologists with each laboratory submission of samples. The standards were purchased from Rock Labs and CDN Resource Labs Ltd, which reflect a range of gold grades that span the grade range at Haile. Since the lab does the sample preparation, and the standard is a pulp, the lab obviously knows that the samples are either blanks or standards. However, they are not informed of the value of the inserted standard or blank.

Drill hole data is initially stored as Excel files at Haile, with each hole reporting the results of the standards, blanks, and duplicates at the bottom of each file. IMC obtained these files and assembled a working spreadsheet of the QA/QC data for statistical analysis. During 2016, OceanaGold loaded data from all primary assay files, including all meta-data and associated QC data, into an acQuire drill hole database.

The 2011 data set that was used for the resource estimate by IMC contained 4,261 standards (not including blanks). This amounts to roughly one standard insertion for every 17 to 18 assay values collected by Romarco drilling during 2011.

Figure 12-1 is a summary plot of the certified sample value on the X axis versus the laboratory reported result on the Y axis. The graph indicates that there a few sample swaps where it is likely that the wrong standard was either recorded or inserted in the sample submission. There are several points on the X-axis where blanks have likely been inserted by mistake rather than standards. This swap rate is acceptable although not ideal.

The graph does not indicate any substantial bias in the results from the project assay lab. The 2011 drill program utilized 36 individual standards with the highest-grade standard (SN50) being 8.68 g/t (0.2533 oz/st).



Source: Independent Mining Consultants Inc.

Figure 12-1: 2011 HGM Standards vs. Certified Value in oz/st

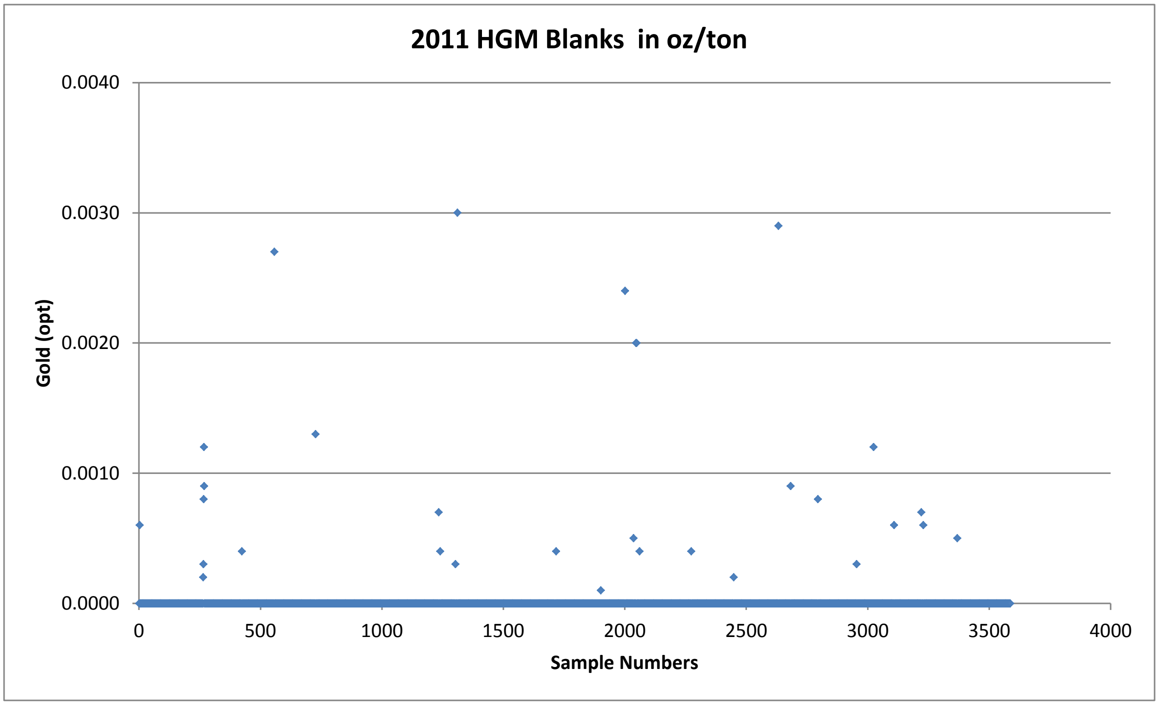
12.2 Statistical Analysis of Romarco Blanks

Blanks are inserted by Haile geologists with each laboratory submission of samples in order to test for contamination. The blanks are purchased from a vendor of materials known to contain no gold. Three types of blank materials were utilized in the 2011 drilling campaign, Marble, Quartz Pebble, and sand.

Drill hole data is initially stored as Excel files at Haile, with each hole reporting the results of the standards, blanks, and duplicates at the bottom of each file. IMC obtained these files and assembled a working spreadsheet of the QA/QC data for statistical analysis.

In summary, the IMC standards data set contained 3,587 blanks (not including standards). This amounts to roughly one blank insertion for every 20 assay values collected by Romarco drilling during 2011.

Figure 12-2 summarizes the results of the blank insertions by sample number. There were 11 occurrences out of 3,587 where blanks were reported as assays greater than 0.034 g/t (0.001 oz/st).



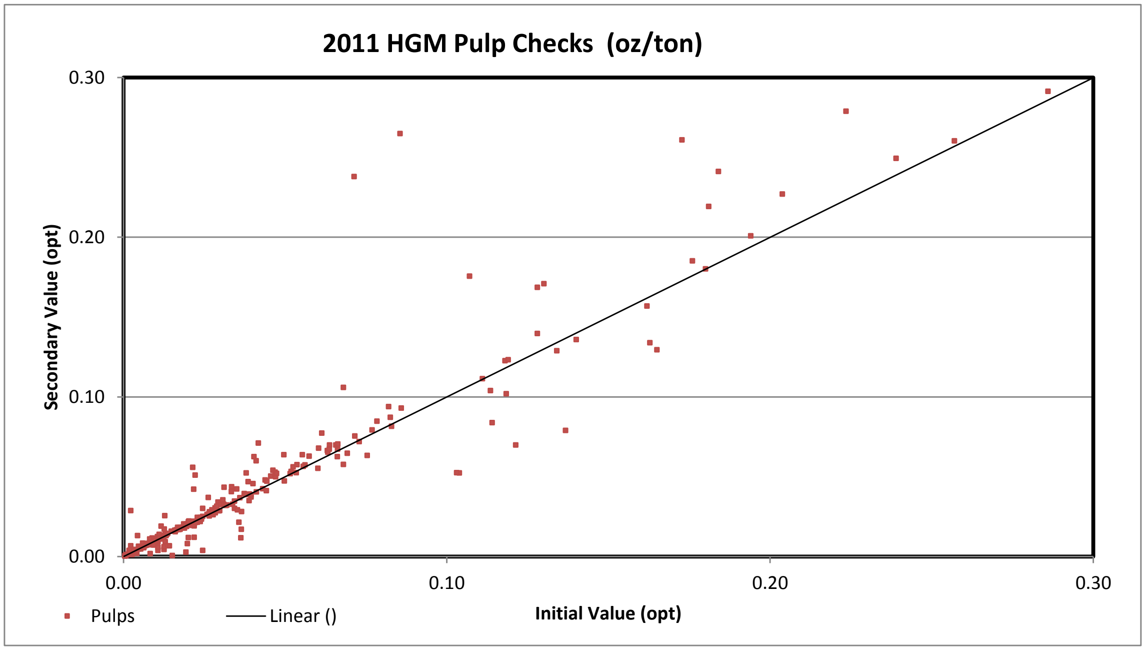
Source: Independent Mining Consultants Inc.

Figure 12-2: 2011 HGM Blanks in oz/st

12.2.1 Statistical Analysis of Check Assays

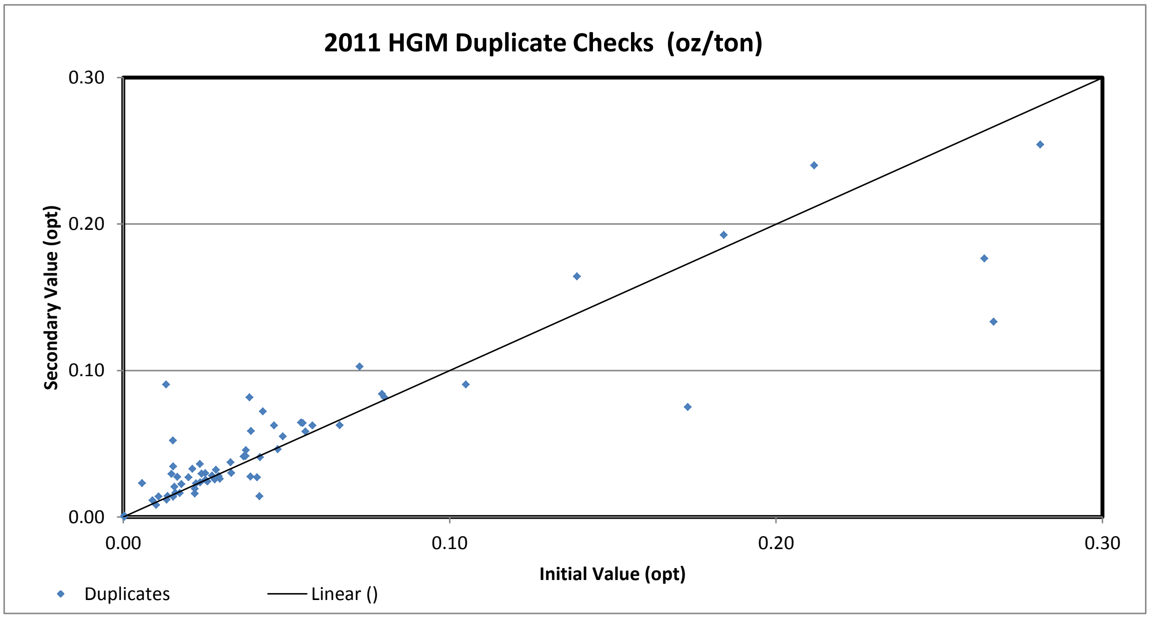
Romarco had consistently sent pulps and duplicates to an outside third-party laboratory. During 2011 this outside check lab was ALS Chemex.

- Pulps were prepared by pulps from AHK and KML that are sent to ALS Chemex as a check on the laboratory analytical procedures.
- Duplicates were ¼ core, or a second split from RC cuttings that are submitted to ALS Chemex for both sample preparation and assay.
- IMC obtained 276 pulp check assays and 76 duplicate results from the 2011 drilling. Figure 12-3 summarizes the results with an XY plot of the AHK and KML assay versus the Chemex check assay on pulps. Figure 12-4 illustrates the XY plot of the duplicate samples.



Source: Independent Mining Consultants Inc.

Figure 12-3: AHK/KML Gold Assays versus Chemex Pulp Assays in oz/st



Source: Independent Mining Consultants Inc.

Figure 12-4: AHK/KML Gold Assays versus Chemex Duplicate Preparation and Assay in oz/st

The Chemex checks average slightly higher than the AHK and KML gold results as evidenced in the range between 1.03 g/t (0.030 oz/st) and 1.71 g/t (0.050 oz/st) on the graph as observed previously during 2010.

The mean of the pulp and duplicate values for fire assay are shown below in Table 12-1.

Table 12-1: Basic Statistics of Pulp and Duplicate Check Assays in g/t

Sample Type	Number of Pairs	AHK/KML Mean	ALS Mean	T Test Result
Pulp	276	3.77	3.91	Pass
Duplicate	75	5.55	6.75	Pass

Source: Independent Mining Consultants Inc.

There was one outlier value of 737 g/t (26 oz/st) within the duplicate checks that was removed from the check statistics by IMC. Values of that level were capped during the block model estimation process as discussed in Section 14.

12.3 Nearest Neighbor Comparison

12.3.1 Romarco Drilling vs Historical Drilling

To gain some comfort with the historical drilling at Haile, IMC completed a nearest neighbor comparison of old drilling versus new drilling on a 6.1 m composite basis. The entire data base of Romarco drilling was used in this analysis rather than just the 2011 component.

The procedure was as follows:

- Drill hole data was composited to 6.1 m down hole intervals.
- Drill holes were tagged with the company that drilled them. In this case, Romarco drilling versus all previous drill holes.
- The data was sorted so that old samples that were within a specified distance of the Romarco composites were selected and paired with the Romarco composite data.
- Only metasediments and saprolite were used in the analysis as they represent the majority of the ore.
- The result is a paired data set where statistical tests can be applied to check that the data represents the same population.

Table 12-2 summarizes the results of the statistical hypothesis test for composites spaced 7.6m (25 ft) and 15.2 m (50 ft) apart. The distances represent one model block and two model blocks respectively.

Table 12-2: Old Drilling Versus New Drilling, Statistical Comparison in g/t and meters

				Hypothesis Tests			
Sample Separation (m)	Number of Pairs (Qty)	New Mean (g/t)	Old Mean (g/t)	T Test	Paired-T	Binomial	KS
7.6	297	6.51	0.92	Pass	Fail	Fail	Pass
15.2	878	0.68	0.82	Pass	Pass	Pass	Pass

Source: Independent Mining Consultants Inc.

The hypothesis tests listed above all indicate that the data could represent the same population with 95% confidence. The purpose of each test is:

- T-Test Comparison of sampled mean values
- Paired-T Comparison of differences between pairs of samples
- Binomial Test that errors are unbiased
- KS Komologorov-Smirnoff test on the overall population

This test did not apply a sort on drill type so that both RC and DDH holes are in the comparison. The comparison of RC vs DDH will be addressed in the next sub-section.

12.3.2 Diamond Drilling vs RC Drilling

The data base at Haile consists of a mix of diamond drilling (DDH) and reverse circulation drilling (RC). IMC compared the results of these two drill methods to confirm that they were not biased relative to one another.

A similar procedure was applied as outlined in the previous section. The 6.1 m (20 ft) composites were coded by drill type, even if both methods were used in the same hole. For example, there are several holes where the top portion was RC drilled, cased, and then deepened with DDH methods.

A nearest neighbor analysis was completed with sample spacing's of 7.6 m (25 ft) and 15.2 m (50 ft). Table 12-3 summarizes the results.

Table 12-3: DDH Drilling Versus RC Drilling, Statistical Comparison in g/t and meters

				Hypothesis Tests			
Sample Separation (m)	Number of Pairs (Qty)	DDH Mean (g/t)	RC Mean (g/t)	T Test	Paired-T	Binomial	KS
7.6	504	0.89	0.86	Pass	Pass	Fail	Pass
15.2	1277	0.89	0.86	Pass	Pass	Fail	Pass

Source: Independent Mining Consultants Inc.

12.3.3 Cyanide Soluble Gold Assays

Early drilling by Cyprus and Piedmont applied cyanide soluble methods to the assay intervals. Much of this effort was directed at measuring the cyanide amenability of the ore to heap leach processing.

IMC completed a comparison between the cyanide data in the historic data base and gold fire assays where they both existed for the same assay interval. There are 9,417 intervals where both cyanide and fire assay data exist. Within those pairs, the cyanide data averages about 67% of the fire assay results. Statistical hypothesis tests do not support commingling of the data.

As a result, IMC chose to ignore the cyanide data within the historic data base and apply fire assay information only to the determination of Mineral Resources and Mineral Reserves.

IMC did complete a test to see if the use of cyanide soluble data could add additional information to the determination of Inferred Mineral Resources. The results could have been potentially conservative, but there was the potential to add tonnage in areas where only cyanide data exists.

The result of the test was that there was no addition of contained inferred ounces with the incorporation of the cyanide data. The low bias in grade offsets any gain that might have occurred in tonnage.

Consequently, the cyanide soluble data was not used in any of the analysis discussed within this document.

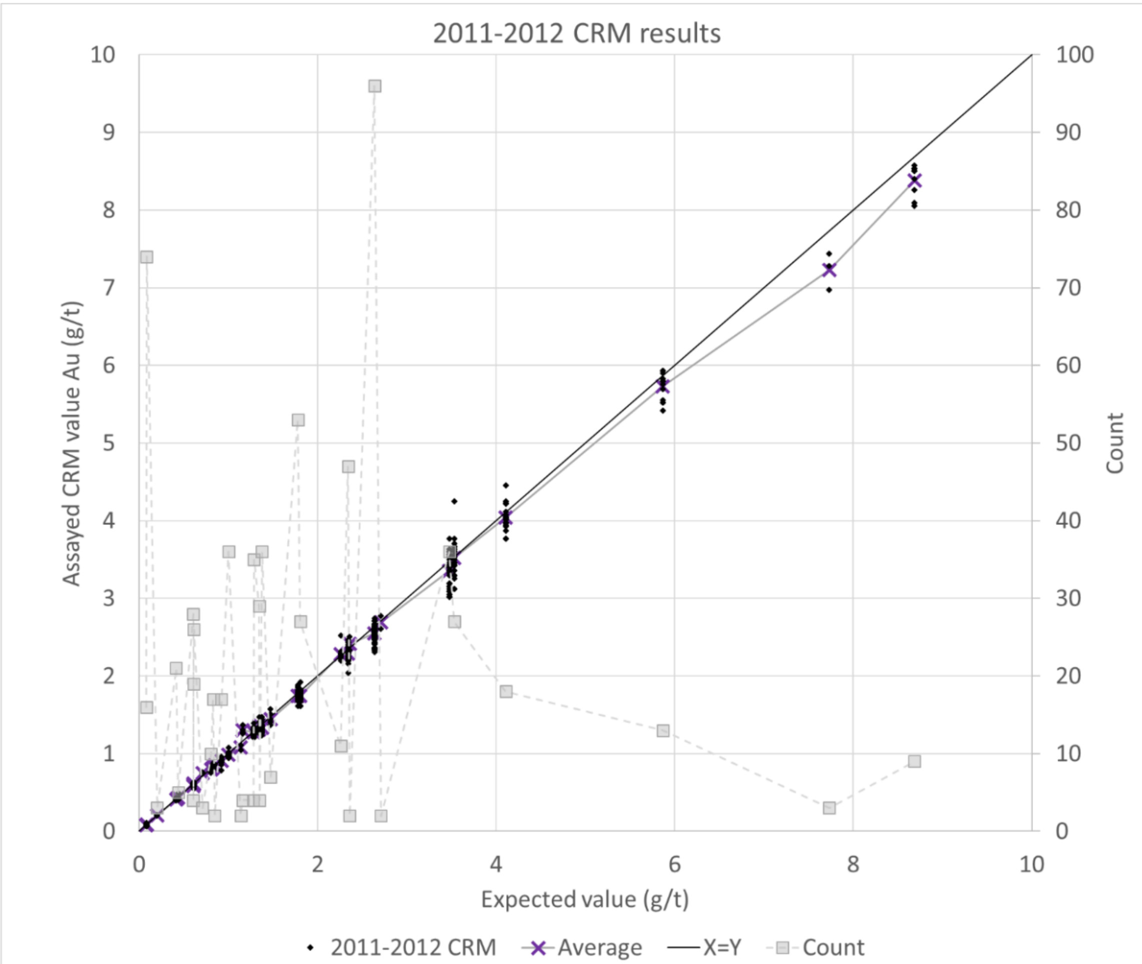
12.4 Haile QC Performance 2011 to 2012

In the period 2011 to 2012, samples were submitted to Alaska Laboratories, Acme Laboratories, ALS and Kershaw Mineral Laboratories (KML). All analysis for Au was by Fire Assay, with either AA or gravimetric finish. Data have been reported (and are stored) in both g/t and ounces/short ton. All data is converted to g/t for resource and reserve evaluations.

12.4.1 2011 to 2012 CRM Performance

Laboratory accuracy was monitored by insertion of commercially Certified Reference Materials (CRM) into the sample stream. A total of 42 different CRM, all sourced from Rocklabs, were submitted in sample batches, for a total of 761 analyses. Only 8 of the 42 CRM had more than 30 insertions. Fifteen samples with obvious errors (sample swaps) were eliminated – a 2% error rate.

Figure 12-5 shows all CRM values plotted against the expected value. Purple crosses show the average of results for a CRM, while greyed boxes show the number of submissions for each CRM. Note that there is no significant bias present below 5 g/t. Above 5 g/t there appears to be slight low bias in analyses relative to expected value, although the number of analyses is low.

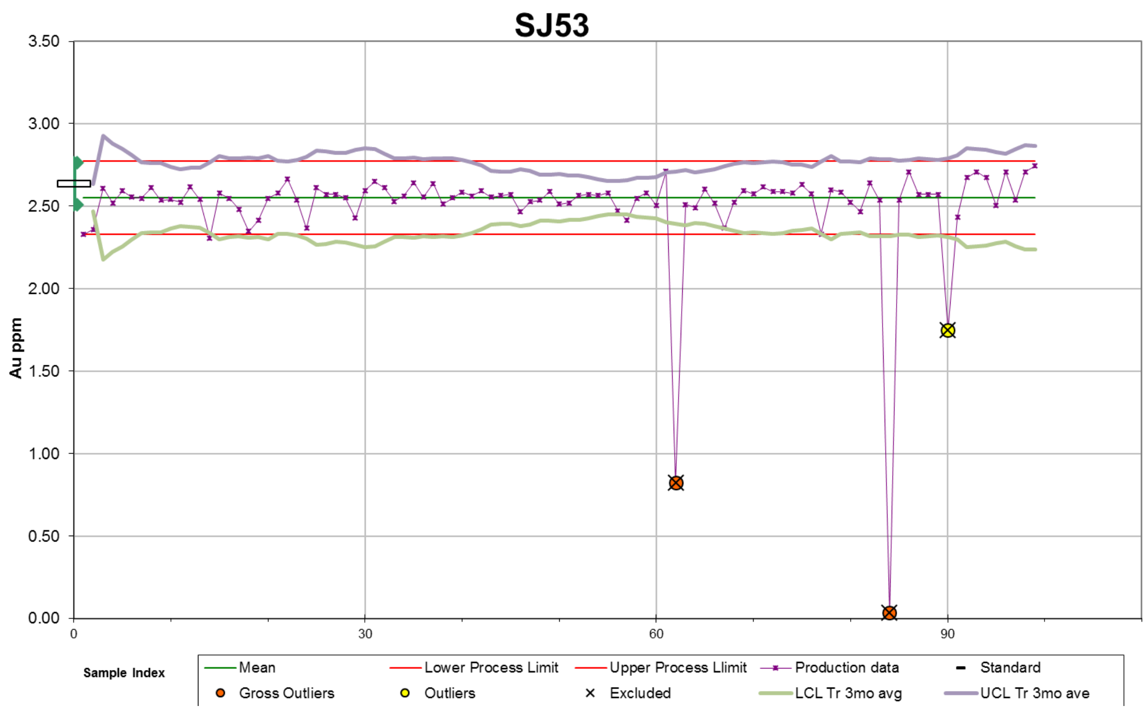


Source: OceanaGold

Figure 12-5: 2011 to 2012 CRM analyses versus expected value

The low submission rate of CRM makes the assessment of temporal stability difficult. Standard control charts were plotted for the CRM with the largest number of results. An example (SJ53) is shown at

Figure 12-6. No obvious trends in the process mean are apparent in any of the longer run CRM results and there are only minor variations in precision. There is nothing in the CRM results to indicate any significant issues with the accuracy of analyses from contributing laboratories in the 2011 to 2012 period.

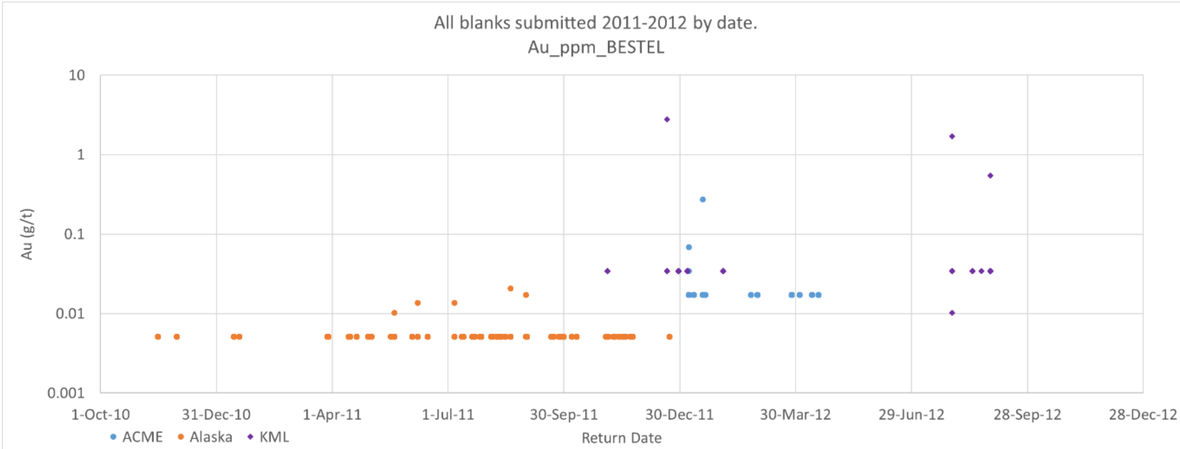


Source: OceanaGold

Figure 12-6: Control Chart, SJ53 March 2011 to August 2012

12.4.2 2011 to 2012 Contamination

Contamination is monitored by insertion of blank materials. In the period 2011 to 2012, a total of 750 samples of three different materials were inserted: Marble, Sand and Quartz Pebble. Results (by laboratory) are shown in Figure 12-7. Four major and one minor non-compliances are observed, two from Acme Laboratories and three from KML. Overall, there is no major indication of contamination during sample preparation. It is possible that CRM may have been mislabeled as blanks during sample preparation.



Source: OceanaGold

Figure 12-7: 2011-2012 Blank Insertions

12.4.3 2011 to 2012 Precision

During 2011 to 2012, a total of 570 pulp duplicates were collected, 509 from Alaska laboratories and 61 from ACME. Unfortunately, the great majority of the duplicates are from very low grade intervals and only 21 of 570 pulps are from intervals where the mean of the pair is more than ten times the detection level. This is too few pairs to obtain a reasonable estimate of the 'average relative error' inherent in sampling and assaying practices. From these few samples, the coefficient of variation is in the range 2% to 7%, which is typical for pulp duplicates.

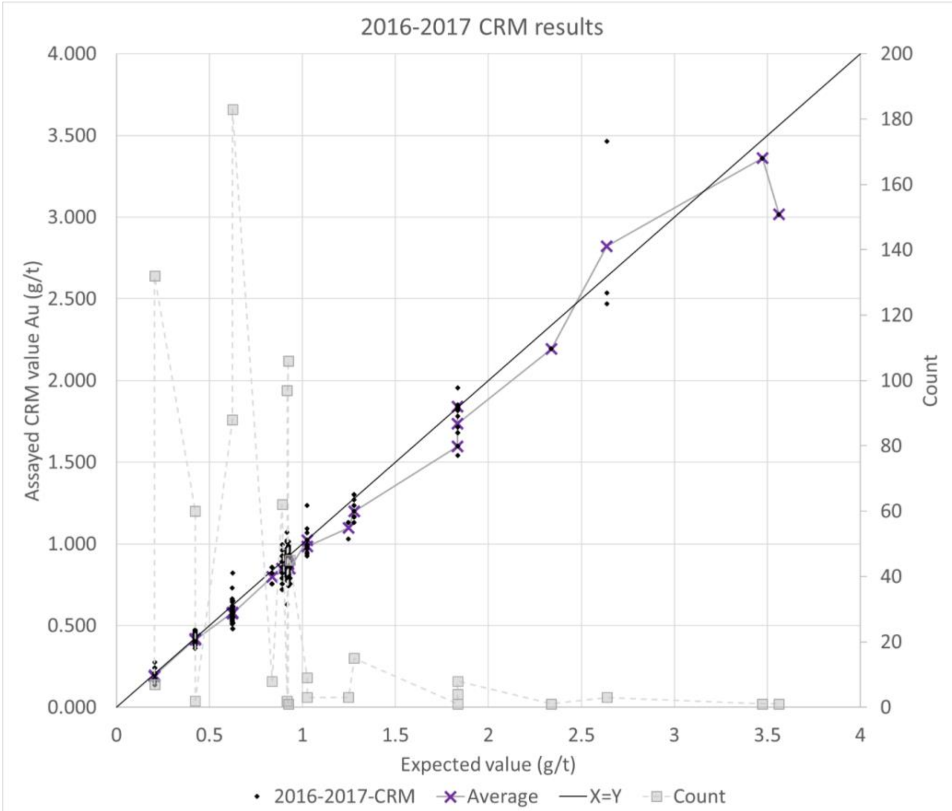
12.5 Haile QC Performance 2016

During 2016 exploration/resource definition samples from Haile drill holes were routinely submitted to KML. ALS was also used as an umpire laboratory.

12.5.1 2016 CRM Performance

Laboratory accuracy was monitored by insertion of CRM into the sample stream. A total of 15 different CRM, all sourced from Rocklabs, were submitted in sample batches, for a total of 848 analyses. Six of the 15 CRM had more than 30 insertions. Eleven samples with obvious errors (sample swaps) were eliminated – a 1.2% error rate.

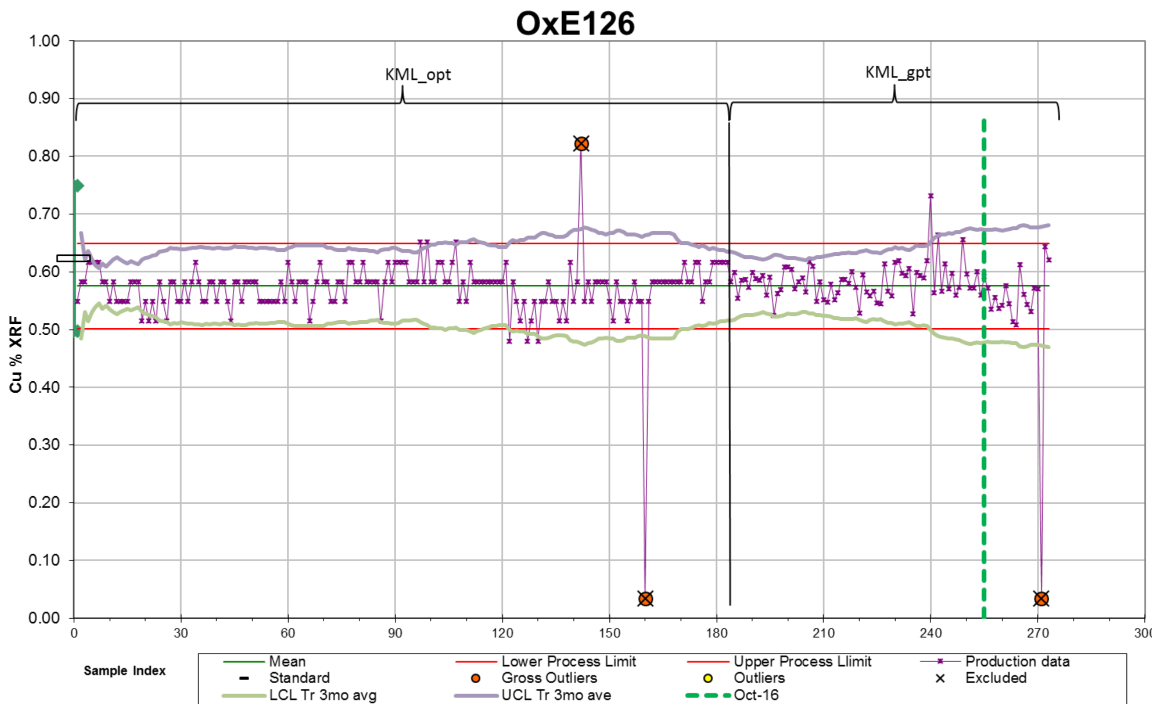
Figure 12-8 shows all CRM values plotted against the expected value. Purple crosses show the average of results for a particular CRM, while greyed boxes show the number of submissions for each CRM. In the six CRM where there are more than 30 results available, there is an appreciable low bias present, ranging from -1.4% to -9.9%. It is likely that reported grades in this range are conservative.



Source: OceanaGold

Figure 12-8: 2016 CRM analyses versus expected value

As an example, the control chart for CRM OxE126 is shown in Figure 12-9. The process mean and control limits calculated from the data are illustrated by the green horizontal line and red control lines. The expected value from certification is shown by the black rectangle on the Y axis, and the expected control limits are shown by the green error bars around that box. In this case, there is a -7.6% difference between the process mean (0.576 g/t) and the certified mean (0.623 g/t). Note that the precision of results changes when reporting changed from ounces/ton to g/t Au in October 2016, as shown on the plot.

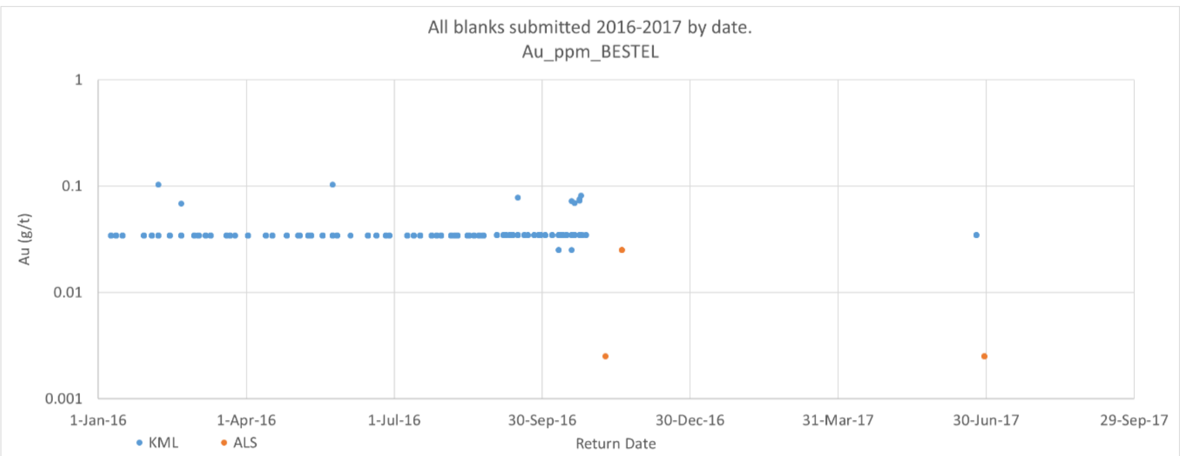


Source: OceanaGold

Figure 12-9: Control chart, OxE126, 2016

12.5.2 Contamination Monitoring

Contamination is monitored by insertion of blank materials. In 2016 a total of 750 samples of three different materials were inserted: Marble, Sand and Qtz Pebble. Results (by laboratory) are shown in Figure 12-10. There is no indication of contamination during sample preparation.



Source: OceanaGold

Figure 12-10: 2016 to 2017 blank insertions

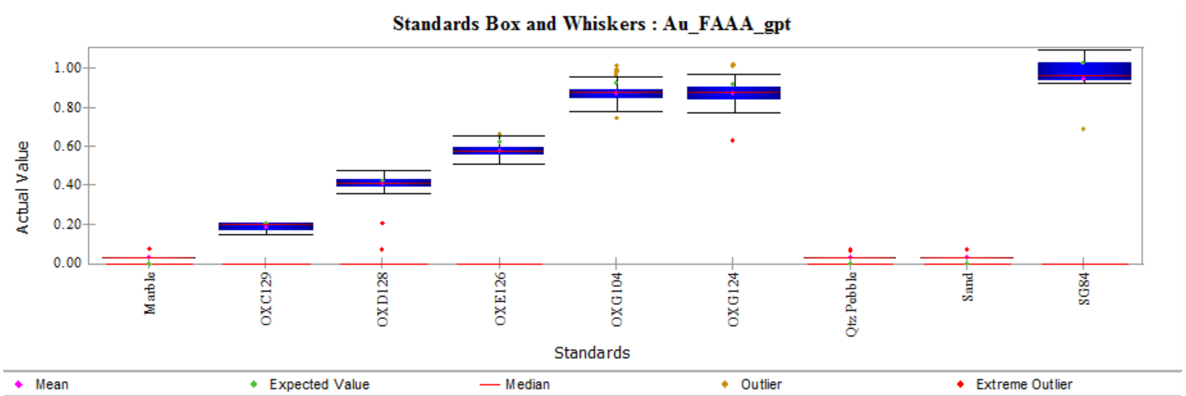
12.5.3 2016 Precision

During 2016 to 2017, a total of 76 pulp duplicates were collected from KML. Only 7% of duplicate pairs are from intervals where the mean grade is greater than ten times the detection level, which does not leave a sufficient number of pairs to calculate a reliable 'average relative error'. From the 19 pairs available, the calculated coefficient of variation is 2%. From the data available, Haile sampling in 2016 appears to meet or exceed industry typical precision for pulps.

12.6 Horseshoe Data Verification 2016

Horseshoe drill hole data were validated by OceanaGold in late 2016. Validation included assays, collar locations, and downhole surveys in the acQuire database for Romarco and OceanaGold drill holes. This includes a 5% check of assay values and a 100% check of collar coordinates and downhole surveys. An analysis of standards and blanks from the 2015 to 2016 Horseshoe drilling program was also conducted. All Horseshoe drilling was core using OceanaGold LF90 drills and company drillers. Sample preparation and assays were conducted by OceanaGold's Kershaw Mineral Lab (KML) at Haile. Check assays were performed by ALS in Tucson, AZ. No significant errors were identified by the study.

Standards and blanks greater than 20% of the expected value were scrutinized and were re-assayed in some cases. Of the 1,699 controls submitted for FA-atomic absorption (AA) during the Horseshoe drilling campaign, 39 standards and blanks (2.3%) are outside acceptable limits. Most failed controls were in unmineralized zones. Validated blanks were used for assays up to 0.080 g/t. Box and whiskers plots for standards and blanks were generated and evaluated as shown in Figure 12-11. Batches with failed standards in mineralized zones were re-assayed at KML and passed the second time.



Source: OceanaGold

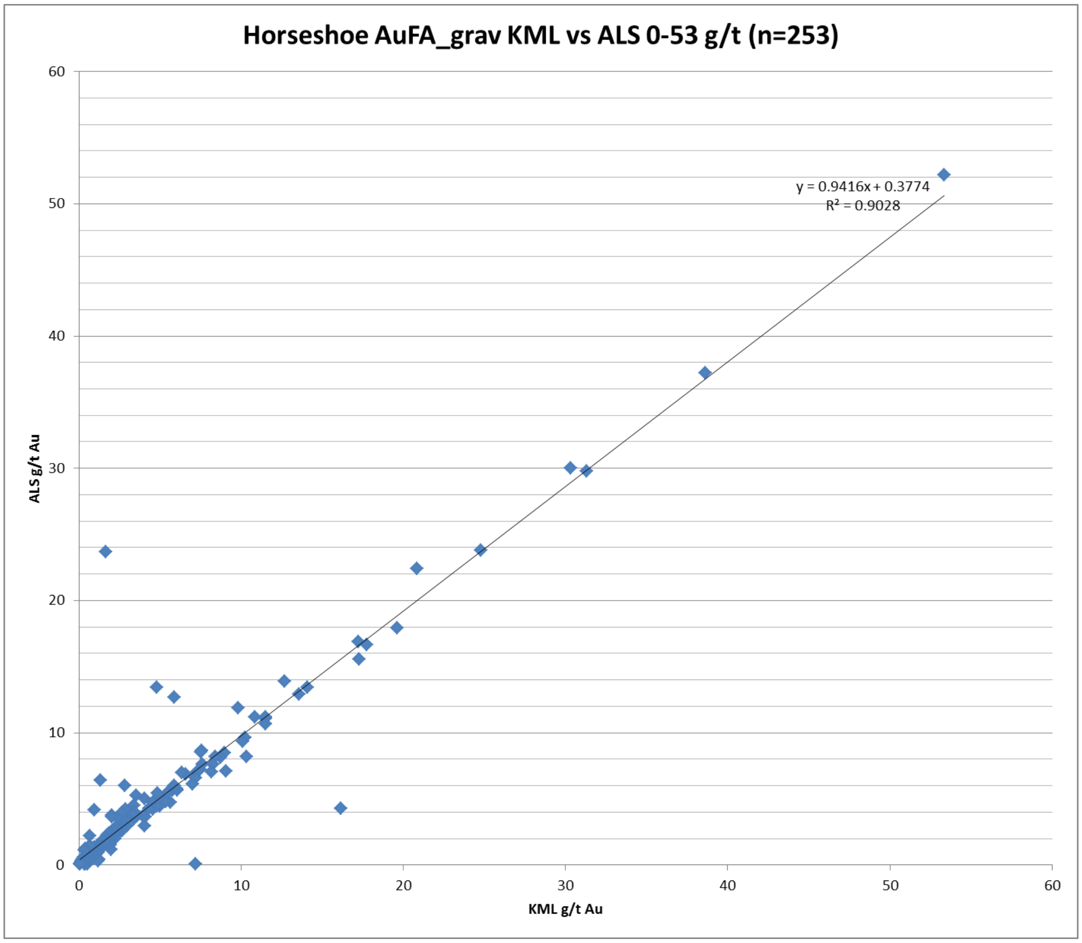
Figure 12-11: Box and whiskers plots for Horseshoe 2016 standards and blanks

A check assay program was conducted for the Horseshoe drilling at ALS Tucson, AZ. Conclusions were:

- KML is 10% lower for AuAA values than ALS for 259 Horseshoe sample pairs in the 0.5 to 3 g/t range (R^2=0.73).
- KML is 5% lower for AuFA values than ALS for 253 Horseshoe samples pairs in the 0 to 53 g/t range (R2=0.90) shown in Figure 12-12 Assays >100 g/t Au (n=18) show poor correlation

when comparing KML to ALS results, likely due to the presence of coarse gold and difficulty in achieving assay precision.

Statistical variance from these studies for AuAA vs AuFA comparisons (n=512) between KML and ALS Tucson indicate that the KML lab is 5% to 10% low, or conversely that the ALS lab is 5% to 10% high. KML adjusted their AA dilution process in October 2016 to achieve better fit with expected values. There were no changes to fire assay procedures at KML. It is suggested that gold assay upgrades of 5% can be applied for internal planning, but that no upgrade be used for external reporting.



Source: OceanaGold

Figure 12-12: Horseshoe AuFA_grav KML vs ALS 0-53 g/t (n=253)

This section focuses on verification of the drilling, sampling, and assaying completed for the data included in the Horseshoe block model completed by OceanaGold Exploration on November 20, 2016.

The OceanaGold database verification for the Horseshoe drilling utilized the following steps:

- Assay Verification – 5% check of assay values.
- Collar Verification – 100% check of collar locations.
- Downhole Survey Verification – 100% check on downhole surveys
- Standard and Blank QA/QC

- KML vs ALS Horseshoe assay comparison

The approach presented above was to verify the reliability of data used in the Horseshoe resource.

12.6.1 Assay Verification

Five percent of samples from 192 holes with gold assay values and located in the Horseshoe model block were selected for verification. Every twentieth sample from the selected holes and the top 50 assay values in the model block, if not included in the original selection, were compared to assay certificates received from various labs. Values of zero or half the detection limit were considered valid for samples below detection. Where duplicates were run, the first reported value was compared with the AuBEST value in acQuire.

This review considered 2,537 samples, with 91% being confirmed to match certificate values. No certificates were located for 175 samples from 15 holes, but database values were consistent when compared to Excel files. For hole DDH0207 and DDH0208 no certificate or Excel file could be located except for one sample from DDH0208 which was consistent with a rerun value. In a small number of cases the original value was close but not equal to the acQuire value. This is likely accounted for by reruns that were not verified during this review.

12.6.2 Collar Verification

Collar locations were verified for 196 holes. This includes four holes for which gold assays were not run, and 1 hole that was not surveyed. Collar excel files were reviewed and surveyed coordinates were compared to the coordinates exported from the acQuire database. All duplicates discovered during the review of collar files were saved into the spreadsheet used for the comparison. An Excel formula was used to confirm matching values and suggest a more detailed review if values were not an exact match. Most discrepancies were accounted for by rounding to the nearest 0.1 or 0.01 ft. Rounding rules were inconsistently but to insignificant effect.

Collar survey files could not be found for four holes. Four coordinate sets did not match, but only coordinates for DDH0537 differed significantly from the acQuire export. Further review of DDH0537 is necessary. The other three holes that did not match likely had duplicates that were imported into acQuire. For RC1754 and RC1757 there were three duplicates, and for unknown reason the one least like the other two was selected. No other significant discrepancies were discovered.

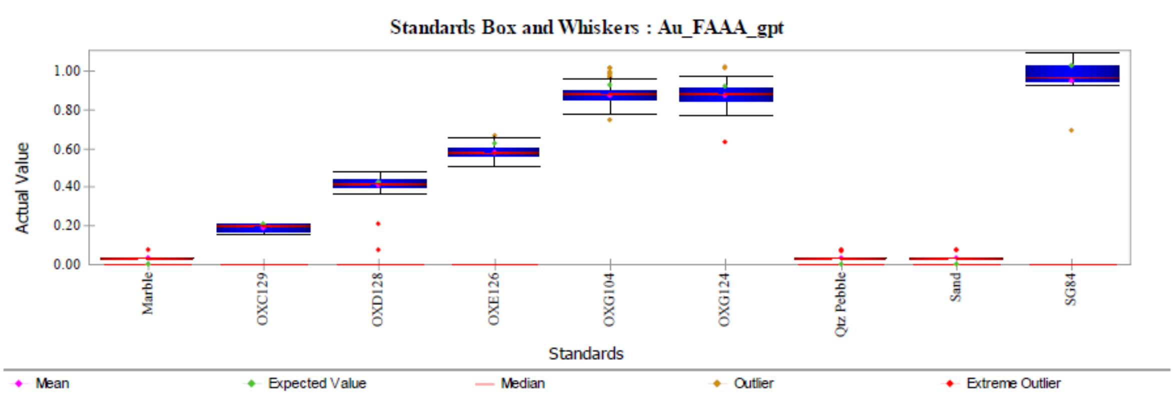
12.6.3 Downhole Survey Verification

Downhole multi-shot surveys or gyroscopic surveys were verified for 195 holes. DDH0576 was not surveyed or assayed as it was abandoned and did not reach its target depth. Downhole survey verification was conducted in two steps. If Reflex exports existed for the relevant holes those files were imported into an Excel spreadsheet along with the downhole survey exports from acQuire. A formula was used to confirm that the data matched. If values did not match in the upper portion of the hole it was assumed that this was due to adjustments related to magnetic interference from casing. Collar azimuths and dips are manually entered as a part of this adjustment process. Holes or portions of holes for which Reflex exports could not be located were copied to another tab and these values were confirmed from paper files or pdfs, or it was confirmed that no survey existed. Adjustments to azimuths (RC1753) and depths (RCT0044) were noted.

Eight survey files could not be located, including the lower portion of RCT0056. It is probable that surveys were not conducted for two of those holes. For DDH0552 and DDH0562 survey files were inconsistent with end of hole depths and survey points exported from acQuire indicating that other files likely exist for those holes. These discrepancies should be further reviewed. Surveys for DDH0528 and DDH0554 had to be adjusted for the lower half of the hole due to instrument error during the survey. For DDH0568 the hole could not be surveyed and an estimated survey was created using orientation tool data and a constant planned azimuth. Surveys exist for DDH0378 and RCT0028 and should be added to acquire.

12.6.4 Statistical Analysis of Standards and Blanks

QA/QC statistics were run in acQuire for controls from all assays received between January 1, 2016 and October 27, 2016. The time period corresponds to the 2015-2016 Horseshoe drilling programs. Re-assayed batches were prioritized over original assays for this analysis if they were imported into acQuire prior to this report. At the time of this analysis standards or blanks outside 20% deviation of the expected value were further scrutinized and were re-assayed in some cases. Of the 1,699 controls submitted for FA-AA during the specified period, 39 standards or blanks are outside acceptable limits (2.3%). Most failed controls are in unmineralized zones and are of no concern. Validated blanks were allowed to assay up to approximately 0.08 g/t. Several other controls are in analytical runs that have been reevaluated but not yet imported into acQuire. Where no reasonable explanation could be discovered the analytical runs containing failed standards and blanks have been resubmitted to KML. Box and whiskers plots for standards and blanks used during the 2015 to 2016 Horseshoe drilling are shown in Figure 12-13.



Source: OceanaGold

Figure 12-13: Standards and Blanks – Actuals vs Expected Values. (Actual Au on Y axis, expected values within the box corresponding with Y axis Au values)

12.7 KML vs ALS Horseshoe assay comparison, Haile

12.7.1 Summary

Three studies were conducted from September to November 2016 to examine possible assay bias at the Haile KML lab by: 1) comparison with standard reference samples; and by checks at the ALS Tucson lab for Horseshoe pulp samples using 2) AuAA methods for the grade range 0.5 to 3 g/t Au, and 3) AuFA methods for all grade ranges. The KML vs ALS assay checks were conducted for the November 2016 Horseshoe data validation and block model process. Results were:

1. KML has a linear 7% low AuAA bias compared to standard reference samples in the 1.7 to 3 g/t range (Vandergriff, Oct. 11, 2016).
2. KML is 10% lower for AuAA values than ALS for 259 Horseshoe sample pairs in the 0.5 to 3 g/t range (R2=0.73).
3. KML is 5% lower for AuFA values than ALS for 253 Horseshoe samples pairs in the 0 to 53 g/t range (R2=0.90). Assays >100 g/t Au show poor correlation when comparing KML to ALS results, likely due to the presence of coarse gold and difficulty in achieving assay precision.

12.7.2 Introduction

Three studies were conducted to investigate assay precision and accuracy at KML (Haile).

1. Third party confirmation assay results in Q3 2016 indicated that KML had a potential low bias. Internal investigation showed that linear 7% low bias versus CRM was prevalent, but that the system was in control because results were within industry statistical allowance (Vandergriff, Oct. 11, 2016). There was no evidence of the bias in accredited proficiency test samples.
2. Comparison of pulp assays was conducted between KML and ALS for 259 samples using AuAA methods for 259 samples in the grade range of 0.5 to 3 g/t Au from the Horseshoe resource.
3. Comparison of pulp assays was conducted between KML and ALS for 259 samples using AuAA methods for 293 samples in the grade range of 0.0 to 917 g/t Au from the Horseshoe resource. This sample set was subdivided into three grade populations to investigate assay variance:
    a. Below detection limit (n=22), 0.1-53 g/t (n=253), and 100-947 g/t Au (n=18).

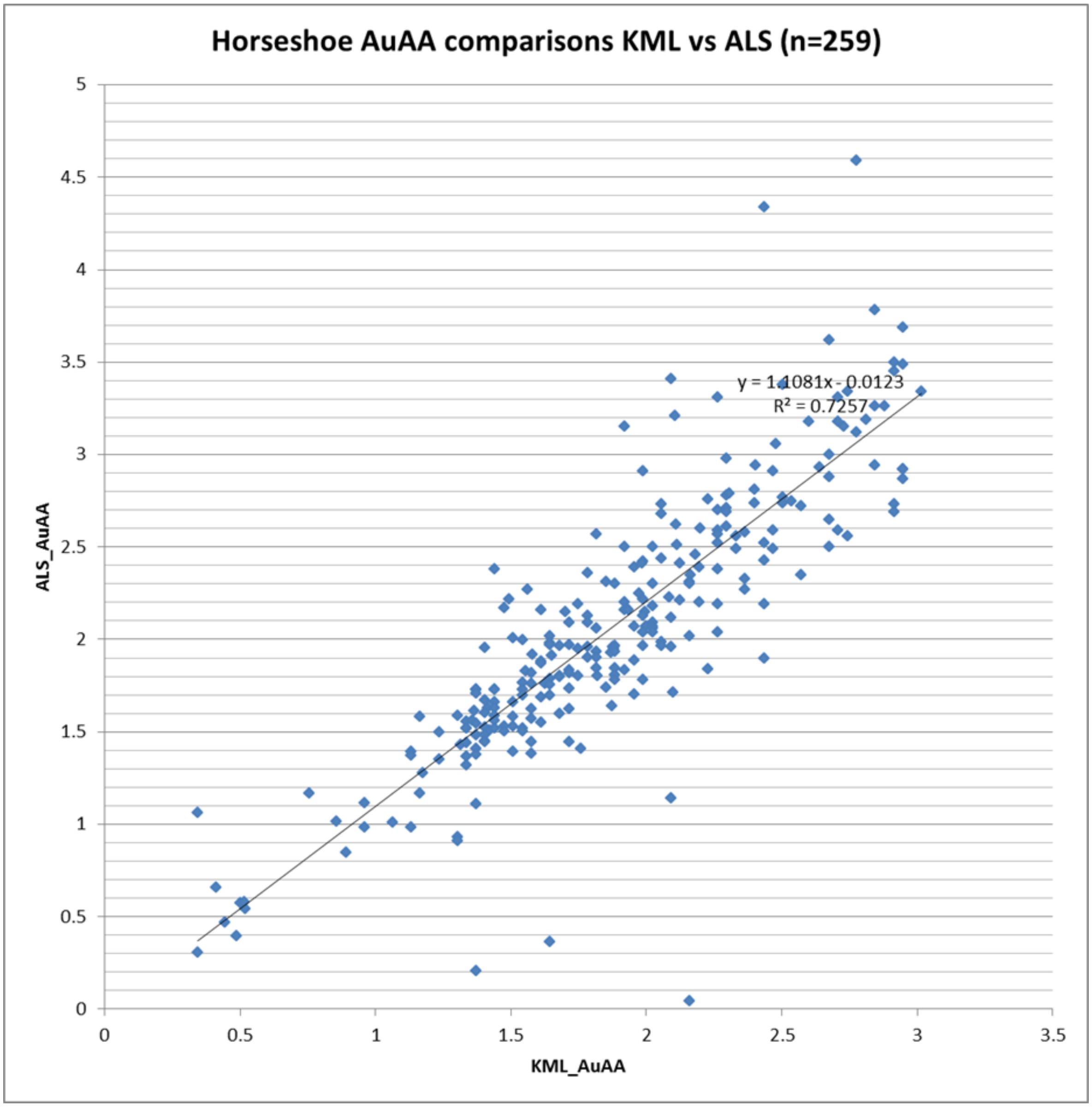
Pulps from the Haile Exploration warehouse were used, and were selected from every 20th interval in mineralized zones >0.5 g/t Au. Check assays were also performed on the 15 highest KML assays. ALS Tucson was selected for check assays due to its ISO/IEC 17025:2005 accreditation and strong industry reputation. The Tucson lab was able to provide quick assay turnaround in two to three weeks for Horseshoe model needs. LECO Corporation (LECO) and multi-element assays were received for 61 elements using ICP-MS methods, but are not discussed in this report.

12.7.3 KML Bias Study

The bias results from calculations performed by the AA are based on extrapolated values instead of values that should be linear. The calibration range of the AA is from 0 g/t to 5 g/t, and should be linear, but the standard dilution procedure forces the determination outside of the linear range and into an extrapolated value that is 7% less than what the concentration should be. This calculation error explains why assay results are so precise, just biased low. Changing the final volume of digested sample to 30 mL and centrifuging the samples has proven to eliminate the issue as it shifts the AA readings into the linear calibration range. The affected data range is only for samples with results higher than 1.7 g/t and less than 3.0 g/t. Less than 1.7 g/t are within the AA instrument's effective range and those greater than 3.0 g/t are analyzed gravimetrically. The bias has been present since analysis at KML began in 2011.

12.7.4 Horseshoe Au AA comparison KML vs ALS

Comparison of assays was conducted between KML and ALS for 259 samples using AuAA methods in the grade range of 0.5 to 3 g/t Au from the Horseshoe resource. All six standards and blanks passed. KML is 10% lower for AuAA values than ALS, as shown in Figure 12-14. Correlation and hence precision is good with R2=0.73. Six samples show very poor correlation. Correlation decreases above 1.5 g/t Au, but is still relatively good.



Source: OceanaGold

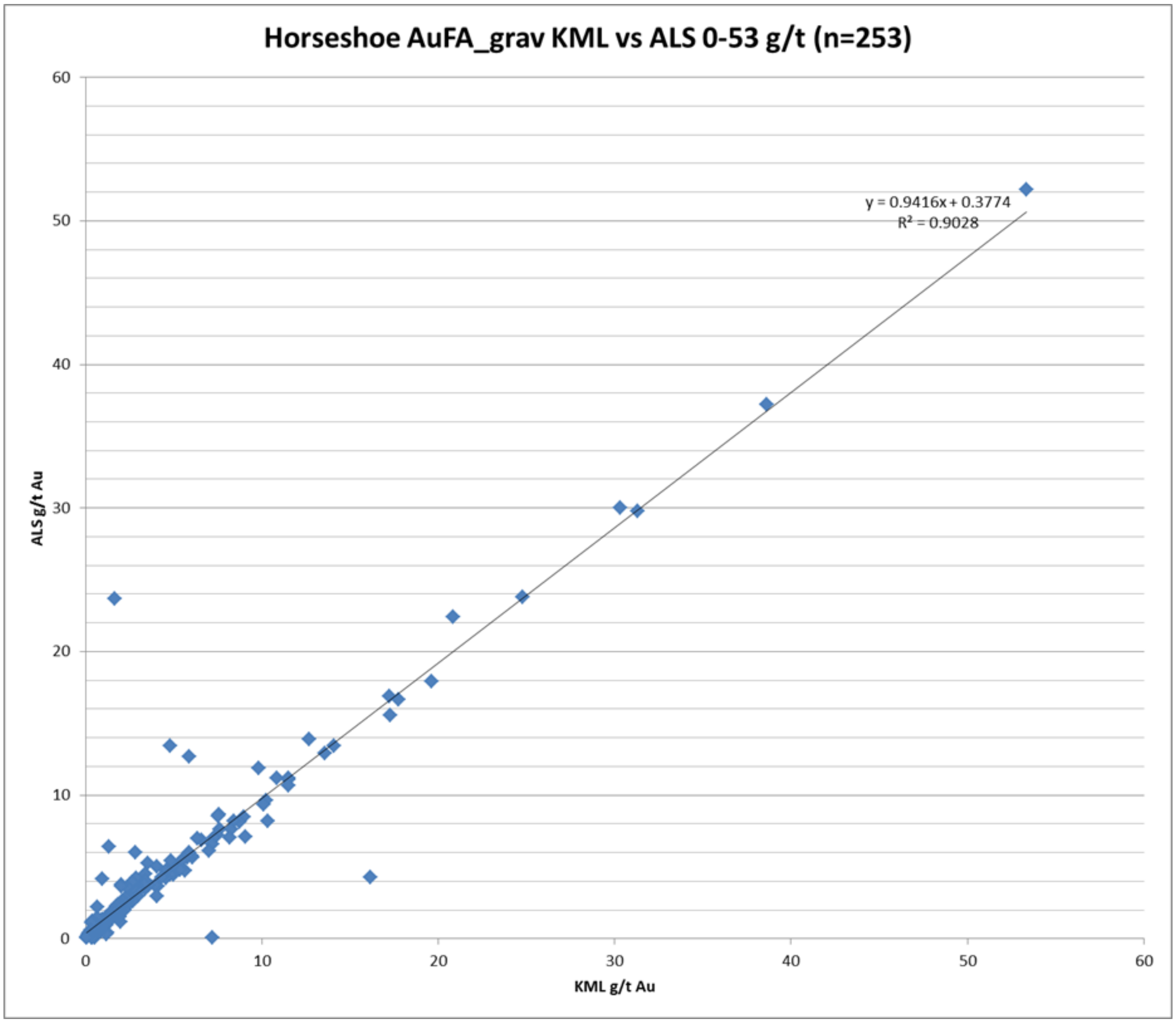
Figure 12-14: Horseshoe AuAA KML vs ALS 0.5-3 g/t (n=259)

12.7.5 Horseshoe Au FA comparison KML vs ALS

Comparison of pulp assays was conducted between KML and ALS for 259 samples using AuAA methods for 293 samples in the grade range of 0.0 to 917 g/t Au from the Horseshoe resource. All six standards and blanks passed. This sample set was subdivided into three grade populations to investigate assay variance and correlation:

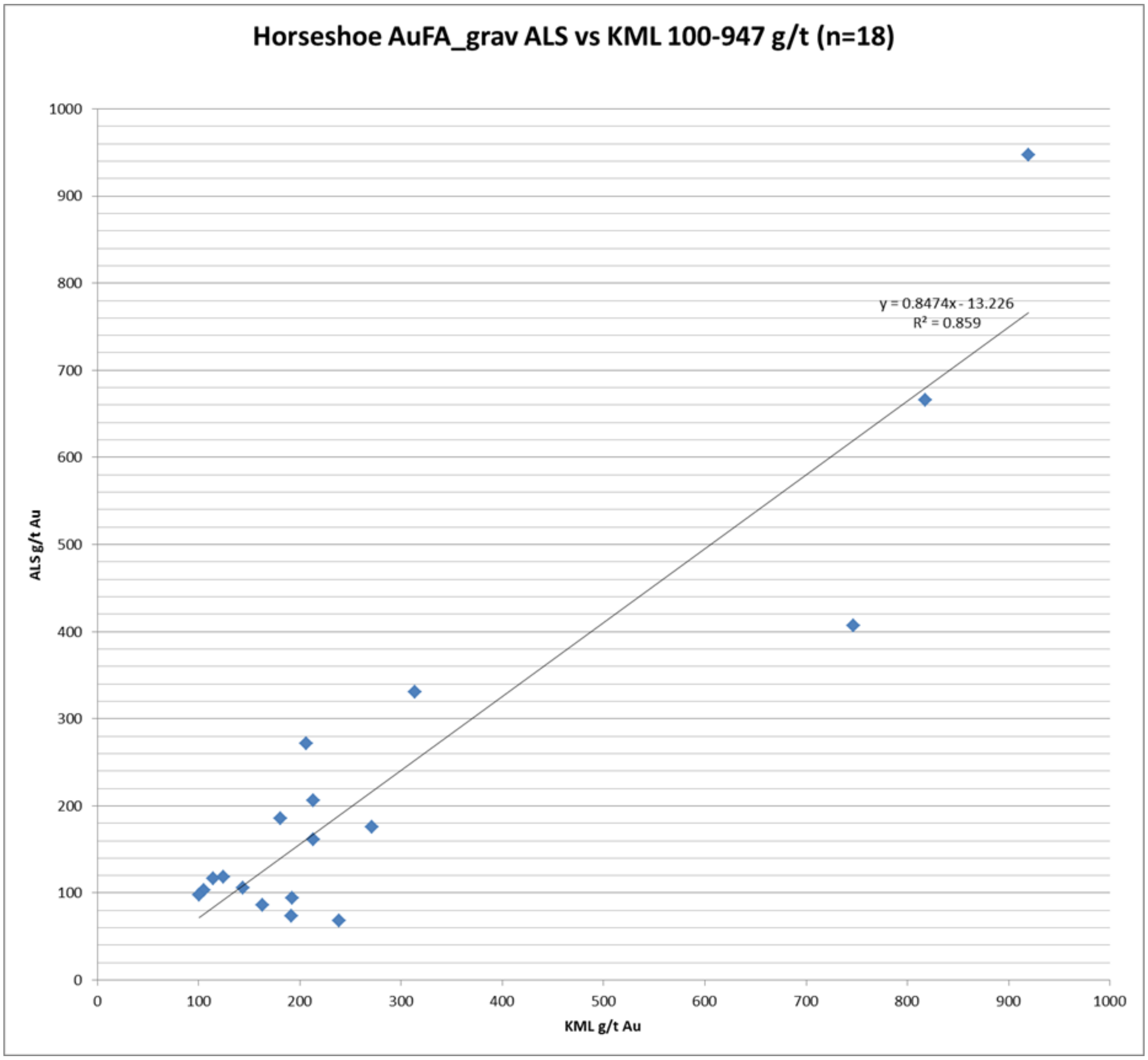
1. Below detection limit (n=22)
2. 0.1 to 53 g/t Au (n=253) (Figure 12-15)
3. 100 to 947 g/t Au (n=18) (Figure 12-16)

KML is 5% lower for AuFA values than ALS for 253 Horseshoe sample pairs in the 0 to 53 g/t range. Correlation is very good with R2=0.90. Three samples show no correlation between KML and ALS at 7, 16 and 23 g/t Au erroneous results are likely. Assays >100 g/t Au show poor correlation when comparing KML to ALS results, likely due to the presence of coarse gold and difficulty in achieving assay precision. Samples below detection limit showed excellent correlation for all 18 sample pairs.



Source: OceanaGold

Figure 12-15: Horseshoe AuFA_grav KML vs ALS 0-53 g/t (n=253)



Source: OceanaGold

Figure 12-16: Horseshoe AuFA_grav KML vs ALS 100-947g/t (n=18)

12.7.6 Conclusions of KML vs ALS study

Statistical variance from these studies for AuAA vs AuFA comparisons (n=512) between KML and ALS Tucson indicate that the KML lab is 5% to 10% low, or conversely that the ALS lab is 5% to 10% high. KML adjusted their AA dilution process in late October to achieve better fit with expected values. There have been no changes to fire assay procedures at KML

13 Mineral Processing and Metallurgical Testing

Sample preparation and characterization, grinding studies, gravity concentration tests, whole ore leach tests, flotation tests and leaching of flotation tailings and flotation concentrate tests were completed to determine the metallurgical response of the ore.

Samples of ore were collected by HGM for metallurgical testing. A series of metallurgical testing programs have been completed by independent commercial metallurgical laboratories. The test work indicated that the ore will respond to flotation and direct agitated cyanide leaching technology to extract gold. The results of the test programs are available in the following reports:

- Phillips Enterprises, LLC (Phillips) 17 September 2008, Progress Report #2 Process and Metallurgical Testing On Haile Gold Mine Ore Project No. 082003;
- Pocock Industrial Inc. (Pocock) Salt Lake City, Utah, May 2009, Flocculant Screening, Gravity Sedimentation, Pulp Rheology, Vacuum Filtration and Pressure Filtration Studies Conducted for Romarco Minerals Haile Gold Project;
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, September 16, 2009, Romarco Minerals, Inc. Haile Gold Project, Metallurgical Report;
- Metso Minerals Industries Inc. (Metso), York, Pennsylvania, December 7, 2009, Test Plant Report No. 20000134-135;
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, March 31, 2010, Romarco Minerals, Inc. Work Index Data for Haile Composite Sample;
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, March 31, 2010, Romarco Minerals, Inc. Metallurgical Testing of Ledbetter Extension Samples;
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, May 27, 2010, Romarco Minerals, Inc. Flash Flotation, Cyanide Destruction & Leaching of Concentrate and Tailing for Haile Composites;
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, September 27, 2010, Romarco Minerals, Inc. Optimization of Leaching of Flotation Concentrate;
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, August 2010, Metallurgical Testing of Horseshoe Zone Samples;
- Metso Minerals Industries, Inc. (Metso), York, Pennsylvania, February 2011, Stirred Media Detritor and Jar Mill Grindability Test on Bulk Flotation Concentrate T11-04;
- KML Metallurgical Services, (KML), Kershaw, South Carolina, December 27, 2012, HGM Years 1 – 3 Silver Characterization Project Test Report;
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, June 6, 2011, Production of Flotation Concentrate and Confirmation Testing of Flowsheet;
- G&T Metallurgical Services Ltd (G&T), Kamloops, Canada November 24, 2011, Flotation & Cyanidation Testing On Samples from the Horseshoe Deposit, Haile Gold Mine KM3076;
- Gekko Global Cyanide Detox Group (Gekko), Ballarat, Australia, July 18, 2016, OceanaGold Haile Gold Mine Cyanide Detox Test Work DTXSC021;
- ALS Metallurgy Kamloops, BC, Canada, December 2016, Comminution Testing on Samples from the Haile Gold Mine KM 5180; and
- ALS Metallurgy Kamloops, BC, Canada, Comminution and Thickening Testing for Haile Gold Mine KM 5293.

The metallurgical test results were used to develop process design criteria and the flow sheet for processing the ore in both the existing and for the upgraded plant.

The following sections contain some information in short tons (st) and others in metric tonnes (t).

13.1 Testing and Procedures

13.1.1 Comminution

Comminution test work on mineralized samples was performed by RDi (using Phillips Enterprises, LLC) and by ALS Kamloops.

Bond ball mill (BM) work indices were determined by RDi for various Haile samples. Bond impact and abrasion tests were also completed. The BM work index results for selected composites from this work are presented in Table 13-1.

Table 13-1: Bond Ball Mill Work Indices (Wi) for Haile Samples

Composite Number	Area	BM Wi @ 100-mesh (kWh/st)
1	Mill Zone	8.42
2	Mill Zone	8.07
3	Mill Zone	7.95
4	Mill Zone	8.03
5	Mill Zone	7.88
6	Haile	8.55
7	Haile	9.78
8	Ledbetter	7.49
26	Snake	10.34
27	Snake	10.39
31	Snake	5.13

Source: OceanaGold

Further testing, including Bond rod mill (RM) index testing as completed on Mill Zone, Haile, Ledbetter and Red Hill ore zone samples. The Bond ball mill work index for each composite was also determined at both 100- and 200-mesh for these samples.

The results for selected composites from this work are presented in Table 13-2.

Table 13-2: Bond Rod and Ball Mill Work Indices for Haile Composite

Composite Number	Sample Description	RM Wi (kWh/st)	BM Wi @ 100-mesh (kWh/st)	BM Wi @ 200-mesh (kWh/st)
2	Mill Zone-Average Grade	11.08	8.21	7.78
6	Mill Zone-High Grade	11.30	8.21	8.17
8	Haile-Average Grade	12.49	9.47	8.92
20	Ledbetter-Average Grade	12.18	8.95	8.42
24	Ledbetter-High Grade	12.56	9.47	9.03
34	Red Hill-Average Grade	-	8.73	9.47
54	Red Hill- Low Grade	-	8.83	9.50

Source: OceanaGold

RDi also performed comminution tests on samples from the Ledbetter Extension zone. The Bond rod and ball mill indices and an abrasion index for an ore composite (83) was determined. The results of this work indices are presented in Table 13-3.

Table 13-3: Rod and Ball Mill Work Indices for Ledbetter Extension Samples

Abrasion Index	Value (kWh/st)
Rod Mill Work Index	12.71
Ball Mill Work Index at 100-mesh	10.21
Ball Mill Work Index at 200-mesh	9.81

Source: OceanaGold

RDi also performed comminution studies on samples from Horseshoe. The Bond rod, ball mill and abrasion indices for four different composites were determined. The samples were relatively abrasive and moderately hard. The results are presented in Table 13-4.

Table 13-4: Rod & Ball Mill Work and Abrasion Indices for Horseshoe Samples

Composite Number	Sample Description (Hole ID / intercepts / lithology)	RM Wi (kWh/st)	BM Wi @ 200-mesh (kWh/st)	Abrasion Index
83	RCT-03 / 1412 to 1460 ft / Silicified Metasediment	-	12.29	0.2167
84	RCT-04 / 1460 to 1510 ft / Silicified Metasediment	-	11.29	0.2691
85	RCT-04 / 1510 to 1585 ft / Silicified Breccia	14.93	12.95	0.3786
86	RCT-04 / 1585 to 1655 ft / Silicified Breccia	13.56	13.77	0.8330

Source: OceanaGold

ALS performed comminution tests on samples from Horseshoe and Ledbetter. The SMC and Bond ball mill indices for composites were determined. The results of this work are presented in Table 13-5.

Table 13-5: ALS Comminution Tests on Horseshoe Samples

Composite Number	A x b	SCSE (kWh/st)	BM Wi @ 200-mesh (kWh/st)
Horseshoe 1	28.9	11.4	13.5
Horseshoe 2	29.9	11.3	13.6
Horseshoe 3	30.7	11.2	9.3
Horseshoe 4	29.4	11.6	10.9
Horseshoe 5	28.0	11.6	14.4
Horseshoe 6	27.1	12.4	10.6
Ledbetter 1	27.3	12.0	11.6
Ledbetter 2	25.6	12.2	13.5
Ledbetter 3	27.8	11.9	11.8
Ledbetter 4	30.8	11.3	8.9

Source: OceanaGold

ALS performed JK Drop Weight and Bond ball mill index tests on samples from mineralized material exposed in Mill Zone Pit. The results of this work are presented in Table 13-6.

Table 13-6: ALS Comminution Tests on Mill Zone Pit Samples

Composite Number	A x b	SCSE (kWh/st)	BM Wi @ 200-mesh (kWh/st)
1a	93.6	7.15	9.4
1b			9.1
2a	52.8	8.85	6.8
2b			6.6

Source: OceanaGold

The limited variability in comminution test work results across different zones indicates that they are similar in terms of ore grindability. Given the common genesis of the assorted zones this is understandable. However, the mineralized material tested that is from deeper zones such as Horseshoe and parts of Ledbetter exhibits increased resistance to comminution.

The comminution circuit design developed for the expansion project has incorporated the characteristics of the material from these deep zones as they do form a significant, albeit minor proportion of the proposed plant feed blend.

13.1.2 Flotation and Cyanidation

The Philips test work described in the September 2008 report was performed on composite ore samples of average grade material from the Haile and Mill Zone pit areas.

The testing was conducted to substantiate metal recoveries from sulfide flotation and cyanide leaching of flotation tailings, and investigate oxidation methods for enhancing gold extraction from sulfide concentrate. Additional work was executed on tailings samples to assess thickening and filtration response, neutralization requirements, and provide material for environmental and tailing disposal engineering studies by others.

The work confirmed the sulfides carry the majority of the metal values in Haile deposits and this allows their concentration into a smaller fraction for processing. Previous operations at the site recognized this and sulfide concentration was practiced. However, the sulfides do not easily release the metal values and limited extraction was experienced by simple cyanidation.

The sulfides contained in the ore composites tested by Phillips showed the same characteristics.

Flotation tests on the Haile composite indicated 66% of the gold was separated into a concentrate that represented 6.7% of the flotation feed mass. Tests on the Mill Zone composite indicated 89% of the gold was separated into a flotation concentrate that represented 13.6% of the feed. The Mill Zone composite test had a finer flotation feed particle size distribution and extended residence time which may explain the difference in recovery.

Leach tests on flotation tail indicated 82% (stage) extraction for gold for both composites. Leach tests on a blend of Haile and Mill Zone flotation concentrate revealed gold extraction of only 67% of the gold with the as-floated particle size. Applying a test procedure entailing a regrind in cyanide solution to 80% passing 15µm, followed by an agitated cyanidation step raised extraction to 80%.

The subsequent phase of work from 2009 was carried out by RDi on samples from the five areas within the Haile Gold mineralized zone; Mill Zone, Haile, Red Hill, Ledbetter and Snake. These discrete areas were provisionally derived from initial distinct open pits but later design work and optimization may make the designation merely a legacy naming convention.

The methodology of compositing samples was to prepare composites from each hole's intervals based on their assays as follows:

- Less than 0.5 g/t Au was considered waste;
- Less than 1 g/t but greater than 0.5 g/t Au were combined as low-grade composites;
- Between 1 g/t Au and 4 g/t Au were combined as average-grade; and
- Over 4 g/t Au were combined as high-grade.

Almost all the samples assayed over 0.3% sulfur and sulfide sulfur accounted for over 95% of the total sulfur.

RDi performed gravity concentration testing using a laboratory centrifugal concentrator with cleaner gravity concentration using a shaking Gemini table. The results indicate that the cleaner stage recovered about 20% of the feed gold but into a concentrate with a mass pull of 1 to 2% of the feed, assaying 11 to 75 Au g/t. The concentrate grade was too low grade to treat separately and there appears to be no coarse gold in the deposit in the ore, thus a gravity circuit was not considered to be applicable.

RDi performed whole-ore cyanide leach tests to examine the effect of ore grind size and leach time on gold recovery. The test work indicated that direct leaching gold extraction from the samples was generally poor and variable, ranging from 40 to 79%.

Most of the gold that leached was in the initial six hours of leach time and extraction generally increased with increasing fineness of grind. The refractoriness of the gold is partially due to size dependence but predominantly due to gold association with sulfides. A summary of the test work is presented in Table 13-7.

Table 13-7: RDi Whole-Ore Leach Test Results

Composite Number	Grind Size (P_{80}, mesh)	% Gold Extraction, Leach Time			NaCN Consumption at 48-hr (lbs/st)
		6-hr	24-hr	48-hr	
Mill Zone Average	100	57.0	65.0	64.7	0.50
Mill Zone Average	200	64.7	65.7	65.9	0.42
Mill Zone Average	325	68.0	69.2	68.4	0.84
Haile Average	200	67.5	71.3	71.5	0.52
Haile Average	325	69.0	73.7	75.3	0.96
Ledbetter Average	200	72.2	75.60	75.8	0.24
Ledbetter Average	325	70.4	80.3	79.1	1.40

Source: OceanaGold

RDi performed flotation test work to investigate the recovery of gold and silver to a sulfide mineral concentrate. The tests indicated that a reagent suite of potassium amyl xanthate (PAX), AERO 404 (or equivalent), and methyl isobutyl carbinol (MIBC) frother, along with a laboratory flotation time of 6-minutes and a grind size of 200-mesh or finer will result in the highest gold recovery values.

A summary of the RDi flotation test work is presented in Table 13-8 and Table 13-9.

Table 13-8: Flotation Test Results – Averages by Grind

Sample Description	Primary Grind (P_{80}, mesh)	Flotation Concentrate 6-minute Flotation Time Recovery %			Concentrate Grade (oz/st)	
		% wt	Au	Ag	Au	Ag
Mill Zone Average	100	18.2	92.7	50.9	0.516	0.341
Mill Zone Average	200	14.2	91.7	58.7	0.630	0.679
Mill Zone Average	325	12.6	90.8	61.6	0.779	0.846
Red Hill Average	200	16.8	82.6	75.2	0.493	1.420
Red Hill Average	325	15.6	82.3	73.1	0.557	1.053
Ledbetter Average	200	10.3	91.8	57.7	1.234	0.749
Ledbetter Average	325	10.5	88.6	42.8	1.301	0.674
Haile Average	200	12.8	86.7	59.9	0.519	0.752
Haile Average	325	11.3	86.4	65.6	0.618	0.834
Snake Average	200	15.4	90.2	50.4	0.665	0.475
Snake Average	325	15.0	91.6	49.0	0.636	0.446

Source: OceanaGold

Table 13-9: Flotation Test Results Average by Grade and Grind

Sample Description	Primary Grind (P_{80}, mesh)	Flotation Concentrate 6-minute Flotation Time Recovery %			Concentrate Grade (oz/st)	
		% wt	Au	Ag	Au	Ag
Mill Zone Average-Grade	200	13.5	93.4	77.1	0.674	1.012
Mill Zone Average-Grade	325	12.9	90.7	70.8	0.697	0.992
Mill Zone High-Grade	200	13.3	92.1	83.5	1.374	1.274
Mill Zone High-Grade	325	12.7	94.8	60.4	1.461	1.015
Red Hill Average-Grade	200	16.6	76.6	83.1	0.338	1.409
Red Hill Average-Grade	325	15.2	82.1	77.8	0.347	0.662
Red Hill High-Grade	200	20.0	93.9	94.3	1.569	3.228
Red Hill High-Grade	325	18.2	93.2	80.5	1.496	2.633
Ledbetter Average-Grade	200	12.2	90.7	68.9	0.703	0.624
Ledbetter Average-Grade	325	14.1	89.5	44.2	0.563	0.271
Ledbetter High-Grade	200	8.0	95.7	57.5	3.071	1.534
Ledbetter High-Grade	325	7.9	87.5	53.3	2.033	1.175
Haile Average-Grade	200	12.2	84.9	65.1	0.365	0.726
Haile Average-Grade	325	11.2	86.5	64.0	0.402	0.682
Haile High-Grade	200	14.8	91.8	86.0	1.595	1.858
Haile High-Grade	325	12.5	87.6	67.3	1.423	1.371
Snake Average-Grade	200	16.4	96.1	53.5	0.472	0.432
Snake Average-Grade	325	17.1	89.1	38.4	0.382	0.350
Snake High-Grade	200	19.0	96.2	69.9	1.575	0.962
Snake High-Grade	325	17.1	95.3	65.6	1.560	0.688

Source: OceanaGold

RDi performed flotation tailing cyanide leach tests to investigate the extraction of gold from the flotation tailing. The test results indicate that gold can be extracted from the flotation tails. A summary of the test work is presented in Table 13-10.

Table 13-10: Flotation Tailing Leach Test Results Average by Grade and Grind

Sample Description	Primary Grind (P_{80}, mesh)	Gold Extraction Leach Time – 24-hr (%)	NaCN Consumption (lbs/st)	Lime Addition $Ca(OH)_2$ (lbs/st)
Mill Zone Average-Grade	200	52.9	0.14	3.08
Mill Zone Average-Grade	325	63.0	0.50	3.08
Mill Zone High-Grade	200	71.7	0.16	3.08
Mill Zone High-Grade	325	71.9	0.44	3.08
Red Hill Average-Grade	200	68.5	0.74	13.19
Red Hill Average-Grade	325	67.5	1.22	12.83
Red Hill High-Grade	200	74.1	2.56	15.76
Red Hill High-Grade	325	81.1	1.40	15.30
Ledbetter Average-Grade	200	68.6	0.44	6.35
Ledbetter Average-Grade	325	70.7	0.24	5.65
Ledbetter High-Grade	200	72.0	0.20	n.r.-
Ledbetter High-Grade	325	76.5	0.16	n.r.
Haile Average-Grade	200	62.7	0.16	13.68
Haile Average-Grade	325	62.2	0.26	13.70
Haile High-Grade	200	75.6	0.22	6.71
Haile High-Grade	325	77.1	0.18	6.31
Snake Average-Grade	200	62.38	0.02	8.53
Snake Average-Grade	325	66.34	0.16	8.45
Snake High-Grade	200	70.00	0.20	6.39
Snake High-Grade	325	70.90	0.24	6.29

Source: OceanaGold

Larger scale flotation test results achieved 91% gold recovery into a concentrate representing 8.8% weight of the flotation feed in 13.5 minutes of flotation time. Subsequent leach tests of flotation tail gave results that indicated 50% gold extraction in 16- hours of leaching.

Regrind test work on concentrate samples generated was performed by Metso Minerals Industries, Inc. (Metso) to predict specific energy requirements for concentrate regrind.

RDi performed flotation test work on twenty-three (23) drill core composite samples from the Ledbetter Extension zone. The methodology for compositing samples by grade was the same as used earlier.

Gold recovery by flotation averaged 86% for the 100-mesh grind samples, averaged 87% for the 150-mesh grind samples, and ranged from 81% to 95% but averaged 89% for the 200-mesh grind samples.

The flotation tailing samples were leached for 24 hours at 40% solids and gold extractions averaged 66% for 100-mesh grind samples, from 52% to 85% and averaged 68% for 150-mesh grind samples, and from 44% to 87% and averaged 69% for 200-mesh grind samples.

In 2010, RDi performed additional metallurgical testing on duplicate ore samples from the earlier testing. Additional composite samples were made to evaluate carbon loading, cyanide destruction, flash flotation, conventional flotation time, and leaching of concentrate and tailing samples.

A procedure was developed and used to evaluate “flash flotation”. Flash flotation was shown to recover 62 to 66% of the gold in two minutes of flotation time. Conventional flotation improves the total flotation gold recovery to about 80% and leaching of flotation tailing extracts 76 to 80% of the gold from the flotation tailing sample.

Fifteen samples were selected for the generation of flotation concentrate in one cubic foot flotation cell tests. The fifteen samples were low, average and high grade from different ore zones (Red Hill, Snake, Ledbetter, and Mill Zone).

Five samples were selected for the generation of flotation concentrate in small scale laboratory flotation cell tests. The five samples were identified as average grade material from the different ore zones.

The flotation tests were followed by leaching tests conducted on the flotation concentrates and flotation tailings. The results of these tests are presented in Table 13-11.

Table 13-11: Test Results for Flotation and Leaching

Test No.	Zone	Grade	Comp. No.	Flotation			Conc. Leaching			Tails Leaching			Total Recovery Au (%)
				Head Grade Au (oz/st)		Au Recovery (%)	Head Grade Au (oz/st)		Au Extraction (%)	Head Grade Au (oz/st)		Au Extraction (%)	
				Assay	Calc		Assay	Calc		Assay	Calc		
1 / 2	RH	L	49	0.027	0.033	91.5	0.172	0.140	62.7	0.003	0.005	83.8	64.5
7 / 8	H	L	47	0.010	0.011	64.7	0.093	0.190	82.6	0.004	0.006	85.9	83.8
17 / 18	S	L	51	0.015	0.015	84.0	0.230	0.245	79.8	0.003	0.003	66.0	77.6
19 / 20	L	L	43	0.021	0.020	86.7	0.248	0.207	71.9	0.003	0.005	61.3	70.5
25 / 26	MZ	L	H290	0.024	0.035	95.4	0.152	0.190	77.4	0.002	0.004	72.5	77.2
15 / 16	RH	A	34	0.080	0.095	92.0	0.589	0.513	83.3	0.009	0.010	67.2	82.0
11 / 12	H	A	8	0.085	0.064	85.5	0.455	0.467	74.8	0.010	0.012	60.3	72.7
9 / 10	S	A	39	0.056	0.052	89.6	0.735	0.583	64.2	0.006	0.006	77.7	65.8
3 / 4	L	A	23	0.059	0.073	89.6	1.009	0.752	80.4	0.008	0.013	71.8	79.5
13 / 14	MZ	A	2	0.057	0.059	92.6	0.423	0.382	69.3	0.005	0.006	69.2	69.3
C34	RH	A	-	0.073	0.072	86.0	-	0.370	80.0	0.012	0.012	80.2	80.0
C28	H	A	-	0.086	0.085	68.1	-	0.580	59.7	0.030	0.029	79.6	66.0
C31	S	A	-	0.051	0.056	93.7	-	0.166	58.5	0.005	0.005	45.1	57.7
C61	L	A	-	0.048	0.047	86.1	-	0.341	80.7	0.007	0.008	81.4	80.4
C5	MZ	A	-	0.073	0.078	92.2	-	0.292	69.5	0.008	0.008	67.0	69.3
27	RH	H	35	-	0.429	94.1	2.601	2.094	73.6	0.030	0.038	77.5	73.8
28	H	H	9	0.180	0.194	90.5	1.394	1.321	88.5	0.021	0.024	64.5	86.2
5/6	S	H	53	0.304	0.312	95.2	2.365	1.875	75.2	0.017	0.020	68.3	74.9
23 / 24	L	H	71	0.240	0.274	94.7	2.622	2.222	74.0	0.015	0.034	81.5	74.4
21 / 22	MZ	H	12/3	0.168	0.199	96.0	1.563	1.155	79.7	0.009	0.020	73.3	79.4

Source: OceanaGold

The overall extraction, sorted by sampled zones is presented in Table 13-12.

Table 13-12: Gold Recovery by Ore Zone and Ore Grade

Ore Zone	Au Extraction – Combined %				Average Au Extraction (%)
	Low Grade	Average Grade		High Grade	
Red Hill	64.5	82.0	80.0	73.8	75.1
Haile	83.8	72.7	66.0	86.2	77.2
Snake	77.6	65.6	57.5	74.9	68.9
Ledbetter	70.5	79.5	80.8	74.4	76.3
Mill Zone	77.2	69.3	69.3	79.4	73.0
Average	**74.7**		**72.3**	**77.8**	**74.3**

Source: OceanaGold

RDi performed additional leach tests on flotation concentrates to ascertain if better results could be obtained.

The results of performing leach tests on larger concentrate samples (i.e., twice the size used in previous tests) demonstrated a significant improvement in gold and silver extraction. Concentrate samples were ground to a size distribution of 80% passing 15 to 18 microns. The slurry was then pre-aerated for four hours and lead nitrate was added for the final three hours of pre-aeration and then leached for 48 hours with carbon present. A summary of the larger scale leach test results is presented in Table 13-13.

Table 13-13: CIL Test Results for Fine Ground Flotation Concentrate

Test No.	Pit	Grade	Composite Number	Grind Size (P_{80}, µm)	48-hr Leach Time % Extraction		NaCN Consumption (lbs/st)
					Au	Ag	
37	Red Hill	L	49	17	80.9	71.1	2.00
36	Haile	L	47	14	77.2	49.5	4.99
38	Snake	L	51	16	81.0	94.4	10.83
35	Ledbetter	L	43	16	88.3	91.9	5.09
21	Mill Zone	L	Hole 290	-	79.8	91.0	5.59
26	Mill Zone	L	Hole 290	-	85.0	82.3	4.75
33	Red Hill	A	34	16	85.8	77.2	4.60
31	Haile	A	28	18	95.6	97.4	4.36
22	Haile	A	8	-	81.6	93.2	3.62
32	Snake	A	31	18	58.8	18.2	4.26
24	Snake	A	39	-	84.7	96.4	5.25
40	Ledbetter Ext	A	61	14	89.8	98.3	1.96
27	Mill Zone	A	2	16	81.5	96.2	4.77
28	Mill Zone	A	5	17	79.2	50.0	4.72
41	Ledbetter Ext	A	73	16	83.7	93.4	3.30
23	Ledbetter	A	23	-	88.3	79.9	4.72
34	Red Hill	H	35	16	92.6	95.9	3.66
29	Haile	H	9	20	93.7	97.7	3.46
39	Snake	H	53	16	83.4	97.4	5.03
30	Mill Zone	H	12/3	19	88.7	95.9	4.00
25	Ledbetter Ext	H	71	-	94.9	95.6	12.3

Source: OceanaGold

KML was commissioned to perform additional flotation and leach tests on 29 composites from Mill Zone and Snake areas. The samples selected were chosen to represent the initial three-years of operation's mine schedule anticipated at the time.

Each composite was subjected to bulk flotation. The flotation concentrate was reground to a P_{80} of approximately 13 microns and leached for 48 hours. The flotation tailing was also leached for 48 hours. The overall gold recoveries ranged from 71.6% to 91.0% and overall silver recoveries ranged from 32.9% to 81.9%. A summary of the results is presented in Table 13-14.

Table 13-14: Tests Results for Composites from Mill Zone and Snake areas

Composite	Au Head Grade (oz/st)	Au Recovery (%)	Ag Head Grade (oz/st)	Ag Recovery (%)
1	0.224	90.6	0.07	75.8
2	0.028	74.7	0.05	74.2
3	0.127	89.8	0.06	73.1
4	0.101	82.6	0.05	73.8
5	0.128	88.6	0.06	81.4
6	0.037	71.6	0.04	68.4
7	0.320	90.7	0.08	77.7
8	0.071	83.7	0.06	77.5
9	0.077	87.9	0.13	81.1
10	0.142	88.7	0.17	81.9
12	0.038	77.5	0.05	64.5
13	0.064	81.3	0.06	78.5
14	0.047	84.0	0.11	80.6
15	0.079	85.1	0.11	75.0
16	0.114	82.6	0.15	75.3
17	0.056	82.9	0.06	78.7
18	0.054	76.0	0.09	75.5
19	0.061	77.7	0.03	70.7
20	0.036	75.8	0.04	76.7
21	0.065	76.6	0.08	71.8
22	0.148	87.5	0.12	71.6
23	0.245	91.0	0.10	74.4
24	0.055	86.5	0.03	52.8
25	0.029	87.6	0.02	43.6
26	0.013	76.3	0.02	62.3
27	0.010	80.8	0.01	58.0
28	0.016	76.8	0.02	35.5
29	0.027	80.9	0.01	32.9
30	0.107	88.8	0.04	46.0

Source: OceanaGold

RDi undertook additional test work on the composite samples from the Horseshoe Zone to determine its response to the process flowsheet selected. Visible gold was reported in some core intercepts used for Horseshoe test work. Test work included comminution (described above) and flotation and leaching of concentrate and flotation tailing.

The flotation process utilizing a simple reagent suite (PAX, AP404 and MIBC) developed for the deposit in earlier studies recovered 85% to 90% of the gold into the concentrate for most of the composites. Cyanide leaching consistently extracted about 70% of the gold in the flotation tailings. The composite concentrate samples were reground and subjected to a preparation step and a cabon in leach (CIL) test, which showed gold and silver extractions of over 90% for most composites. Lower recoveries were achieved for the low-grade composite 83.

A summary of the test results is presented in Table 13-15.

Table 13-15: Test Results for Horseshoe Samples

Composite Number	Assay Head Au (g/t)	Primary Grind Size (P_{80}, mesh)	Flotation Recovery (%)		Tailing Leach (%) Extraction		Concentrate Leach Extraction (%)		Overall Extraction (%)	
			Au	Ag	Au	Ag	Au	Ag	Au	Ag
83	1.8	100	86.5	63.0	70.2	3.7	69.3	64.9	69.4	42.3
83		150	89.0	57.1	70.4	9.6			69.4	41.2
83		200	87.6	54.1	73.9	5.5			69.9	37.6
84	9.1	100	88.4	74.4	69.4	5.3	96.2	94.6	93.1	71.7
84		150	90.2	77.3	71.3	22.0			93.8	78.1
84		200	90.1	77.7	71.0	7.1			93.7	75.1
85	10.4	100	83.3	70.7	66.5	39.4	96.0	91.9	91.1	76.5
85		150	89.2	81.4	71.7	19.1			93.4	78.4
85		200	87.4	80.7	76.3	37.5			93.5	81.4
86	12.1	100	86.4	74.5	67.2	8.9	94.9	92.1	91.1	70.9
86		150	85.8	73.8	72.5	45.8			91.7	80.0
86		200	90.4	75.2	76.6	40.7			93.1	79.4
87	10.6	100	69.7	57.7	59.5	53.4	95.2	95.0	84.4	77.4
87		150	77.8	64.8	66.1	54.1			88.7	80.6
87		200	75.9	69.0	70.8	54.1			89.3	82.3

Source: OceanaGold

In late 2011, G&T Metallurgical Services was selected to perform the metallurgical test program on additional Horseshoe samples. The metallurgical test program involved testing of twelve variability samples to evaluate recoveries by flotation and cyanidation of concentrate and flotation tails.

The Horseshoe samples responded very well to the Haile flowsheet. A summary of the test results compiled by HGM personnel is provided in Table 13-16.

Table 13-16: Test Results for Horseshoe Samples

Composite Number	Gold Head Grade (oz/st)	Flotation P_{80} (µm)	Kinetic Flotation Recovery (%)	Bulk Flotation Recovery (%)	Regrind P_{80} (µm)	Concentrate Leach Extraction (%)	Tailings Leach Extraction (%)	Overall Gold Recovery (%)
1	0.199	81	93.9	86.5	15	90.4	81.1	89.2
2	0.339	70	92.5	90.7	13	99.5	67.3	96.6
3	0.043	78	96.5	94.0	14	86.1	87.6	89.8
4	0.060	73	93.3	97.0	16	98.5	74.1	86.1
5	0.076	81	90.2	83.4	10	94.9	85.5	94.4
6	0.168	84	88.4	86.0	11	93.0	90.2	94.9
7	0.082	82	85.2	88.0	15	94.9	75.6	93.8
8	0.094	66	89.0	84.4	12	98.6	83.3	92.6
9	0.057	69	86.2	89.2	11	97.1	81.7	96.0
10	0.121	75	75.1	77.6	15	90.3	82.1	90.6
11	0.349	88	86.2	87.5	14	97.4	88.4	95.8
12	0.129	77	85.3	83.6	10	97.4	84.2	96.1

Source: OceanaGold

Laboratory testing on ore composite samples demonstrated that the mineralization was readily amenable to flotation and cyanide leaching process treatment. A conventional flotation and cyanide leaching flow sheet can be used as the basis of process design. The relative low variability of flotation test work indicates that the mineralized zones are relatively similar in terms of mineral composition, and flotation and cyanide leaching response.

The samples tested responded favorably at a moderately fine feed size range of 80% passing 200-mesh (74 microns). Therefore, a primary grind size of 80% passing 200-mesh was recommended for process circuit design development. Operational experience may allow this criterion to be relaxed reducing comminution requirements and increasing plant capacity.

The flotation testing indicated that gold can be recovered in a flotation concentrate that will also contain the majority of the silver in the ore.

The tailing from the flotation circuit can then be processed by cyanide leaching to recover gold onto activated carbon.

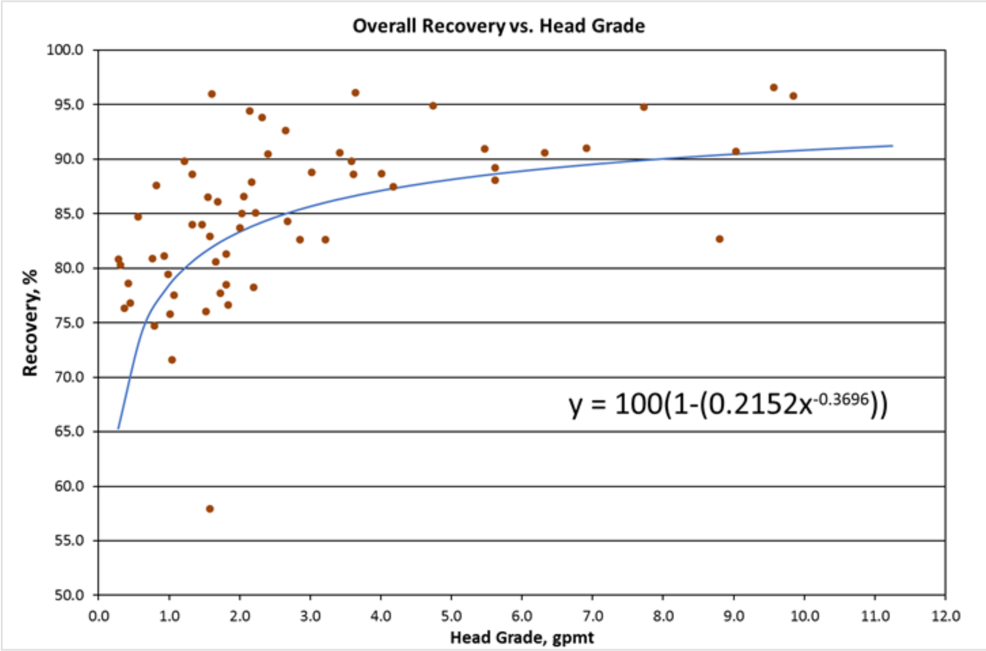
The test work indicated that the circuit should include regrinding of the flotation concentrate before slurry pre-aeration, and a leach time of 24 hours. A regrind circuit product size of 80% passing 15 microns is an appropriate target for regrind circuit design. Operational experience may allow this criterion to be relaxed reducing comminution requirements and increasing plant capacity.

Leaching of the flotation concentrate can extract 82% to 91% of the gold and 80% to 96% of the silver. Leaching of the flotation tailing can extract 45% to 86% of the gold in the flotation tailings. It appears that overall gold recovery will be in the range of 65% to 92% dependent primarily on head grade to the mill and less dependent on from which zone the ore is mined.

The unit operations that determine the amount of gold extraction are flotation, flotation concentrate regrind and leaching, and flotation tailing leaching.

The data developed in the test programs has been used to establish a relationship between overall gold recovery and head grade as shown in the graph in Figure 13-1. The algorithm for the “best-fit” line that describes the head grade to recovery relationship can be used to estimate gold recovery from a predicted mill head grade for example, at a mill head grade of 2.3 g/t, the recovery equation graph predicts a gold recovery of 84.2%.

The results of grade and recovery data analysis are shown in Figure 13-1.



Source: OceanaGold

Figure 13-1: Overall Percent Recovery vs. Head Grade

Overall the results from Horseshoe tests were in-line with or exceeded the recovery model.

The cyanide destruction test results indicated that the sulfur dioxide (SO_2)/air cyanide destruction process destroys weak acid dissociable (WAD) cyanide very effectively, as well as free cyanide. Operations to date have achieved target levels of cyanide removal from the TSF and reagent usage is being optimized to further reduce costs.

13.1.3 Sample Representativeness

Samples were collected from a range of locations across the main area of the resources than is planned to be fed to the processing plant over the LOM.

The minimal variability of test work indicates that the different mineralized zones are relatively similar in terms of ore grindability, chemical and mineral compositions, and flotation and cyanide leaching response. Given the uniform geological setting and mineralization this is not surprising.

It is evident from the comminution testing that ore competency is increased in the deeper resources e.g., Horseshoe underground. This is typified by the trend of increased Bond Ball Milling Work indices for samples sourced from lower levels.

13.2 Recovery Estimate Assumptions

The recovery estimate is based on the interpretation of test work results from a range of samples and test work campaigns. The processing plant is operating in the final stages of commissioning and entering the ramp up phase and as such there is insufficient data to validate the model at the time of issue of this report. However metallurgical recoveries are broadly in line with those forecast using the model.

The product from Haile is gold containing moderate amounts of silver. The doré bars are 95% pure with minimal or no deleterious elements. Mercury has not been detected in assays on any test work.

14 Mineral Resource Estimate

Separate block models were generated for the open pit and then underground areas. As such, the open pit and underground Mineral Resources are described separately.

The open pit block model is “HGM16” built by Bruce Van Brunt, Technical Services Manager, OceanaGold: Haile Gold Mine. Statistical analysis of the drill hole data was completed using Snowden’s Supervisor software. The geologic model was provided by the OceanaGold Haile Exploration department. The topographic surface was the only part of the model modified by SRK using Vulcan software to reflect the most accurate up to date topographic surface. The grade estimation and block model construction was done entirely within Vulcan software. Resource pit shells were generated using Whittle software.

The underground model was built by Bart Stryhas, Principal Resource Geologist at SRK.

14.1 Open Pit Mineral Resources Estimate

14.1.1 Drill hole Database

Cut-off date on the drill hole data utilized in the HGM16 estimate is August 30, 2016. The cut-off date for data utilized in the IMC15 estimate was November, 17 2011. There were 4,139 collars reported from the drill hole database used for HGM16 estimation although not all of those contained mineralized intervals.

Additional drilling since the IMC estimate:

November 17, 2011 to 2012:

- **RC:** 63 holes, total of 22,796 m. Drill Areas: Bayberry (regional property near site), condemnation drilling of Ramona OSA, Robert OSA, JPAG, Lubas, Mill Zone, and Mustang
- **DDH**: 56 holes, total of 27,693 m. Drill Areas: Bayberry, condemnation drilling of Robert OSA, Lubas, Mustang, Palomino, Small, and Snake
- **RCD**: 70 holes, total of 40,462 m. Drill Areas: Bayberry, condemnation drilling of Robert OSA, Horseshoe, Mustang, and Palomino

2013 to 2014:

- **RC**: 45 Holes, total of 2,386 m. Drill Areas: Piezometer and pumping wells in Ramona OSA, Mill Zone, Ledbetter, Small, Snake, diabase holes in Mill Zone
- **DDH**: 23 holes, total of 12,461 m. Drill Areas: Bayberry, Loblolly, and Jackson (regional properties near site)

2014 to 2015:

- **RC**: 14 Holes, total of 143 m. Drill Areas: Piezometer and monitor wells in Mill Zone

2015 to 2016:

- **RC**: 43 Holes, total of 3,172 m. Drill Areas: Monitoring and dewatering wells in Mill Zone
- **DDH**: 9 holes, total of 2,621 m. Drill Areas: Loblolly

2016 to Current:

- **RC**: 10 Holes, total of 798 m. Drill Areas: Piezometer and monitor wells
- **DDH**: 70 holes, total of 23,145 m. Drill Areas: Horseshoe and Palomino

14.1.2 Geologic Model Concepts

Discussions with James Berry, Ken Gillon and Reid Mobley and John Jory about the controlling features of the geology on gold mineralization formed the basis of the approach. In addition to gold grade, the key indicators to gold mineralization are silicification and pyrite content. Core logging has also identified brecciation as a feature often found with high-grade mineralization.

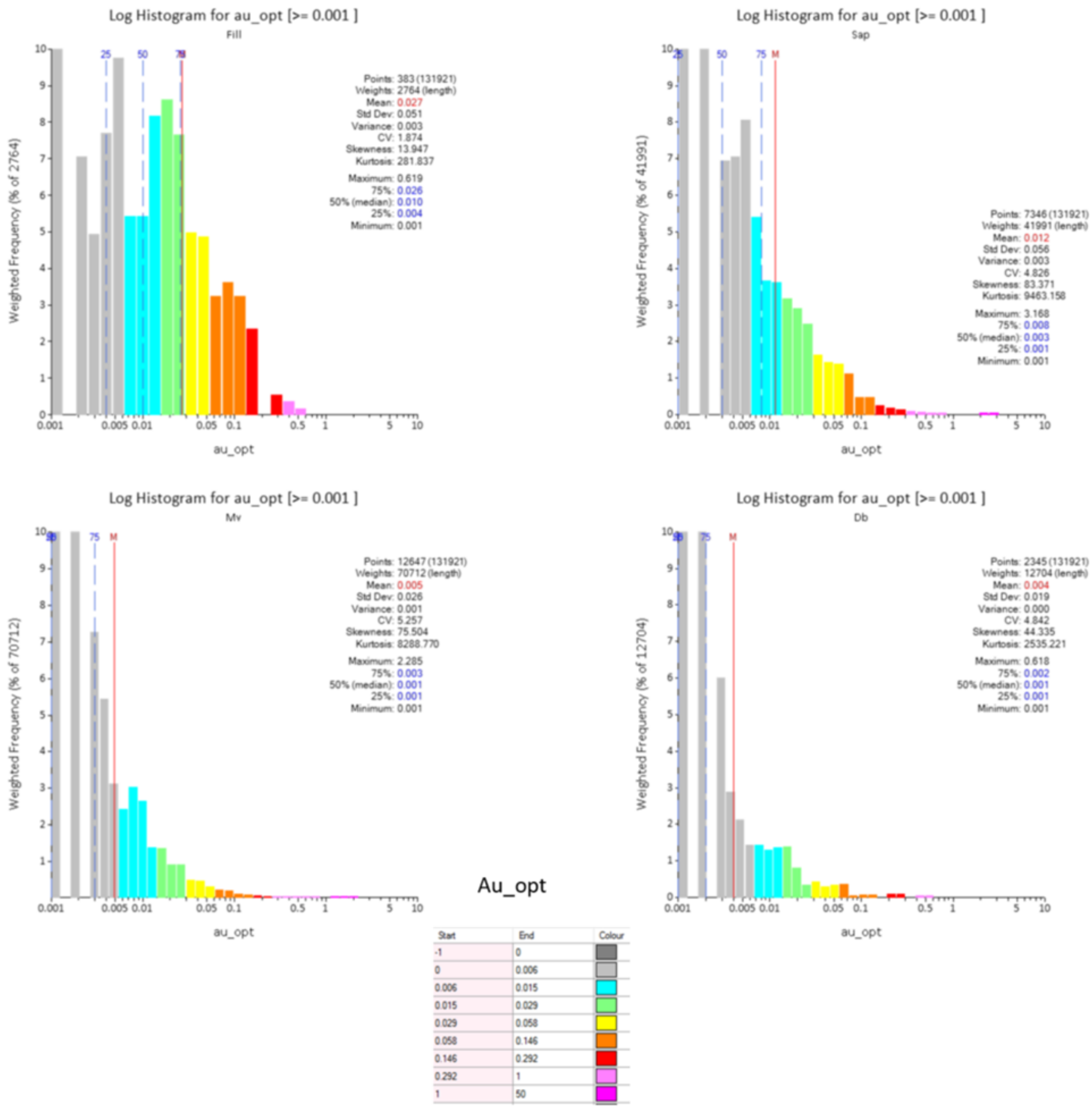
14.1.3 Statistics

Lithologic codes used at Haile in the core shed capture a great many geologic attributes including the primary rock type, presence of brecciation, silicification, lamination and numerous variations on the general rock unit. For the purpose of block modelling and building a resource model these 150+ codes were reduced to a summary set of codes. The following listed codes are deemed to be sufficient for the purpose of assigning pit slopes and to segregate similar types of rock that may have common characteristics such as specific gravity.

- S - sand
- Sap - saprolite
- MV - metavolcanics
- DB - intrusive dikes
- Fill - back-fill from historical mining
- Ml - laminated metasediments
- Ms - silicified metasediments
- Breccia - brecciated rocks

Statistical Analysis of Lithology vs. Au grade was completed and is shown in the following figures.

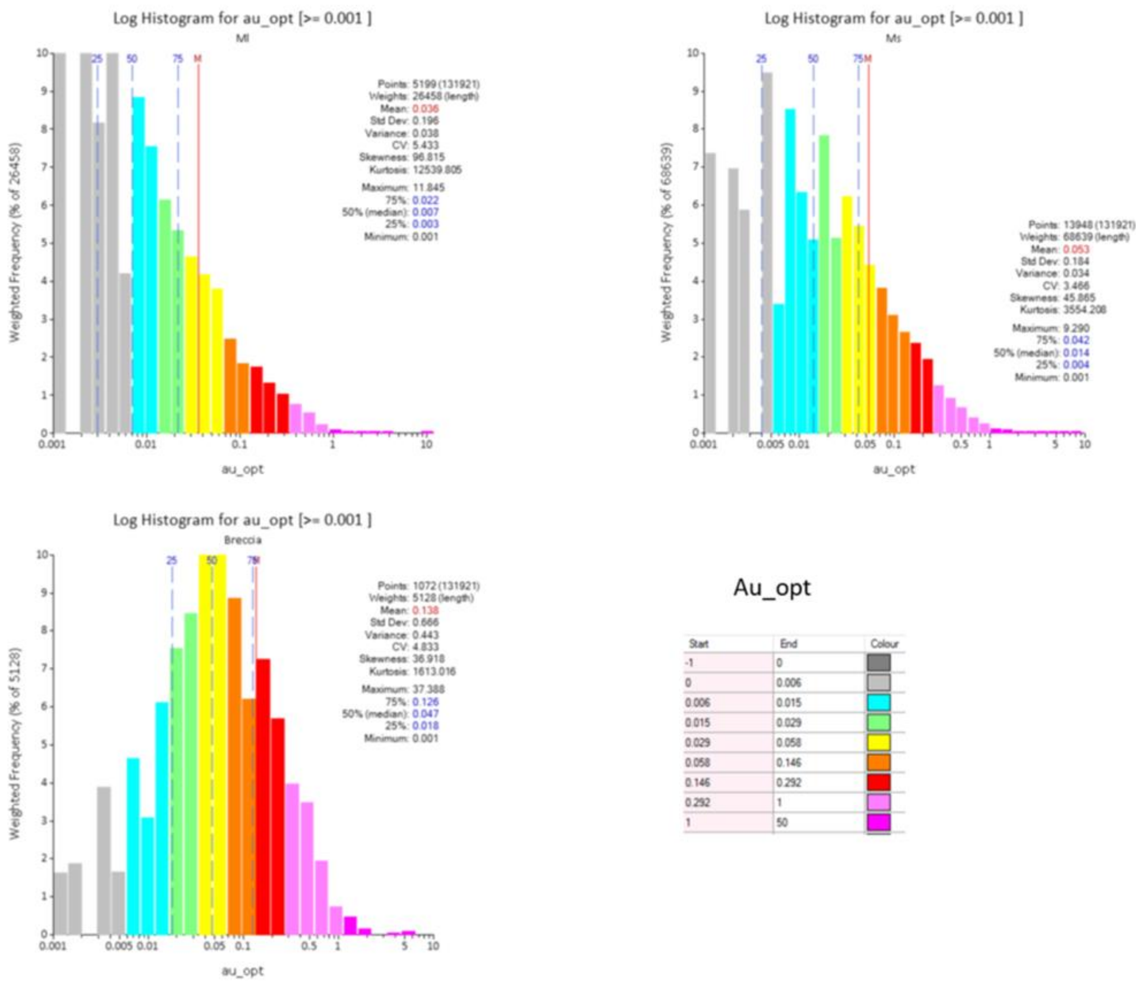
Figure 14-1 shows the statistical analysis of essentially barren rock units and fill vs Au grade (oz/st).



Source: OceanaGold

Figure 14-1: HGM16 Statistical Analysis of Lith code vs oz/st: Barren Rock Units and Fill

Figure 14-2 shows the statistical analyses of mineralized units vs Au grade.



Source: OceanaGold

Figure 14-2: HGM16 Statistical Analysis of Lith code vs oz/st: Mineralized Lith Units

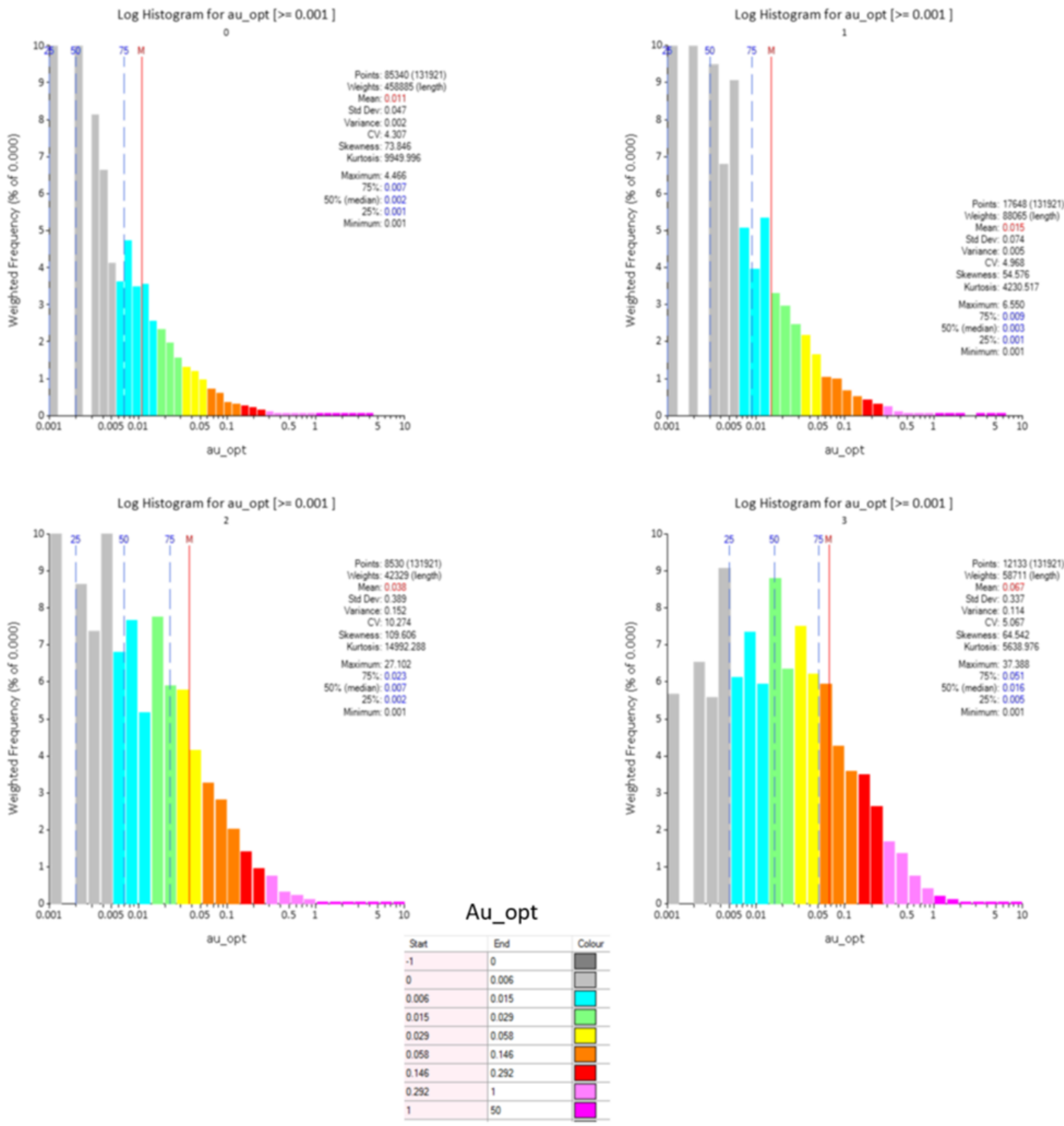
The Ms coding reports higher grade mineralization relative to the laminated Ml unit due to the presence of silicification, however the variance of the two data sets is nearly the same. This suggests the utility of using the silicification as a guide to identifying mineralization.

The 'Breccia' rock type consists of any unit that includes a code denominating brecciation. Although the breccias are allegedly of limited spatial extent, they certainly represent a high-grade component of the mineralization.

14.1.4 Silicification

The progression of 'silicification' increases from 0 (non-existent) to 3 (main component). The minor silicification (1) population has an average grade of about 0.5 g/t. The average grade of moderately silicified (2) rocks is 1.0 g/t and the very silicified (3) average grade increases to approximately 2.0 g/t. The gold to silicification relationship is strong.

A statistical analysis of silicification vs. Au grade in Figure 14-3 below.



Source: OceanaGold

Figure 14-3: HGM16 Statistical Analysis of Silicification code vs oz/st

Any amount of silicification is potentially indicative of mineralization. High-silicification may be a good control of the high-grade gold population during interpolation.

Spatially, the correlation of Silicification = 3 intercepts and Breccia intercepts is quite good as shown in Figure 14-4 and Figure 14-5 shows the spatial analysis of (Silicification = 3 intercepts).

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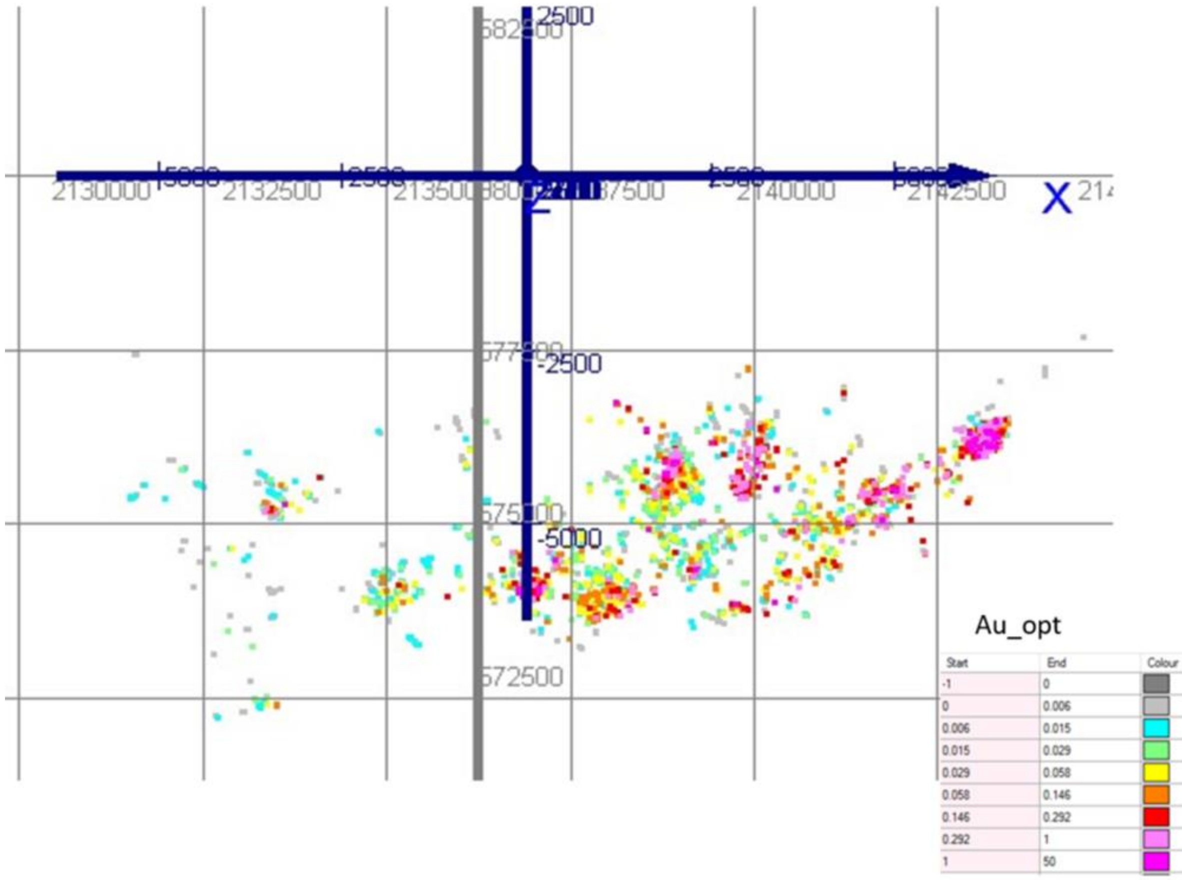
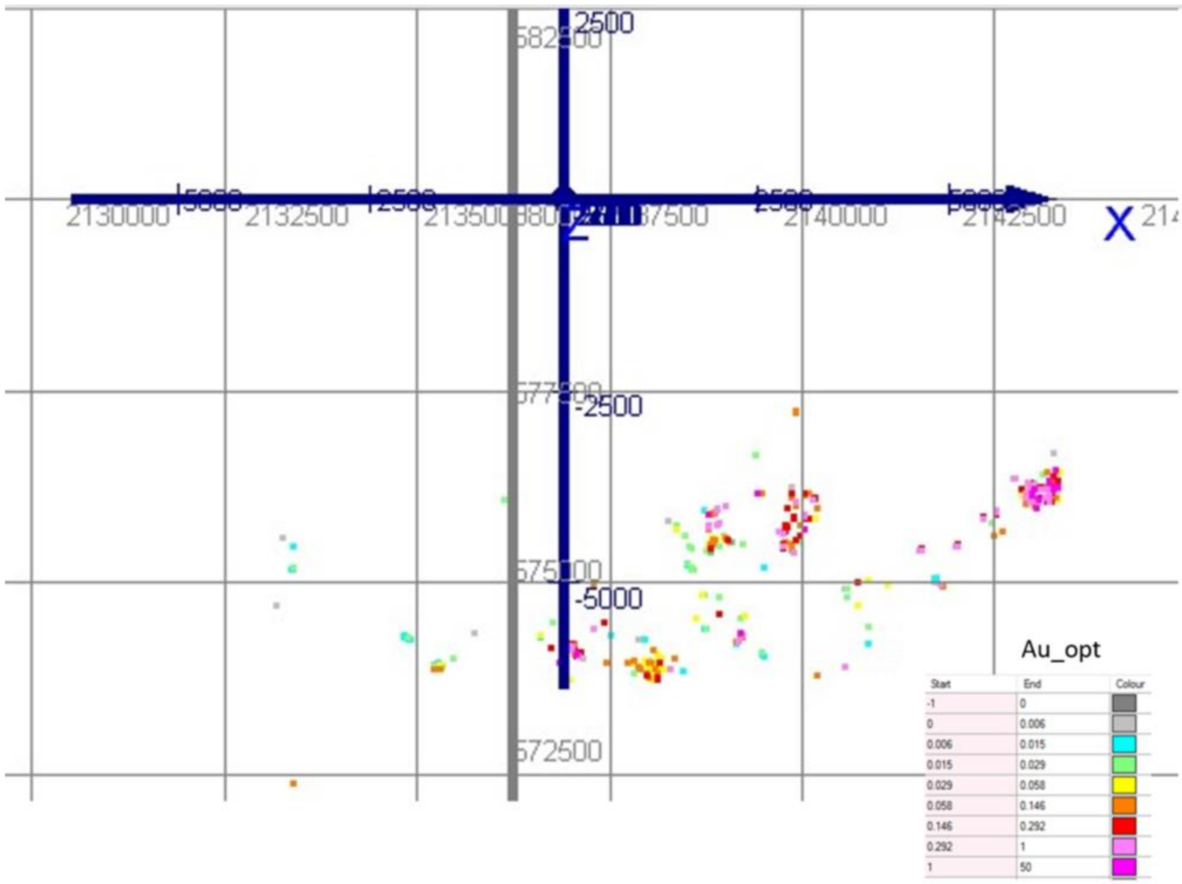


Source: OceanaGold

Figure 14-4: HGM16 Spatial Analysis of (Silicification = 3 intercepts)

Figure 14-5 shows the spatial analysis of Brecciation.



Source: OceanaGold

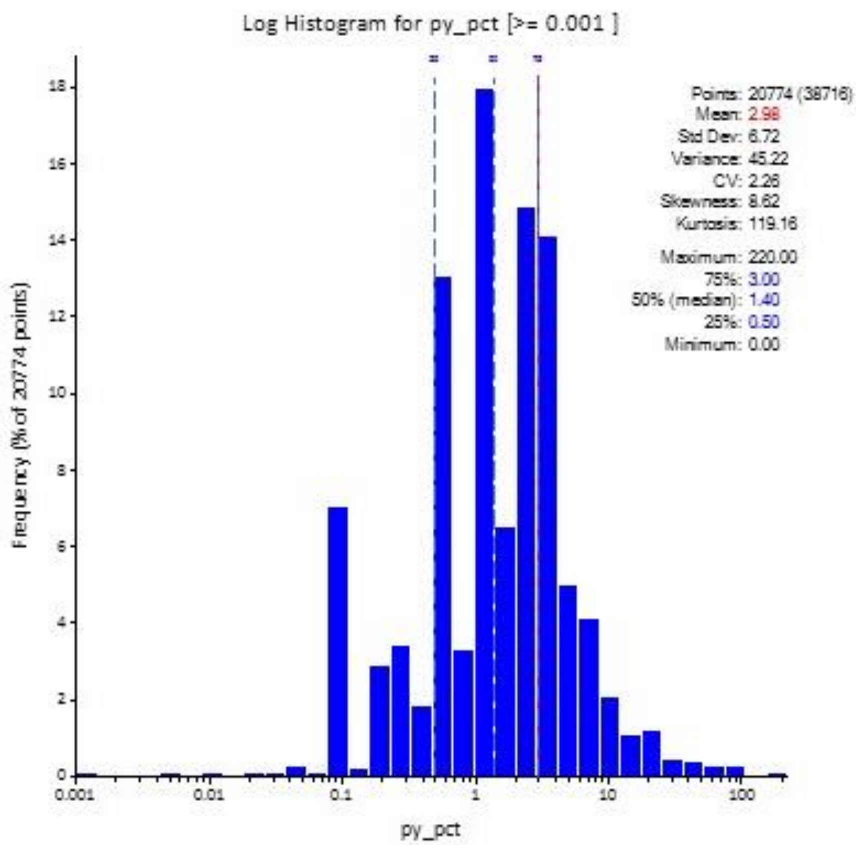
Figure 14-5: HGM16 Spatial Analysis of Brecciation

Potentially, the Breccia zones may lie at the core of, or be otherwise interrelated to, the higher intensity silicified domains. Statistically this inference is supported by the fact that the grade of the Breccia population effectively doubles relative to the Silicification = 3 population however the variance remains proportional.

14.1.5 Pyrite

Multiple morphologies of pyrite have been identified at Haile, ranging from fine to coarse cubic pyrite. Based on logging it has been established that the fine-grained pyrite is commonly associated with mineralization.

Filtering out the lower gold grades below 0.2 g/t gives an indication of the association between pyrite and gold mineralization. Figure 14-6 shows that when gold is above 0.2 g/t, on average 3% pyrite is also present.



Source: OceanaGold

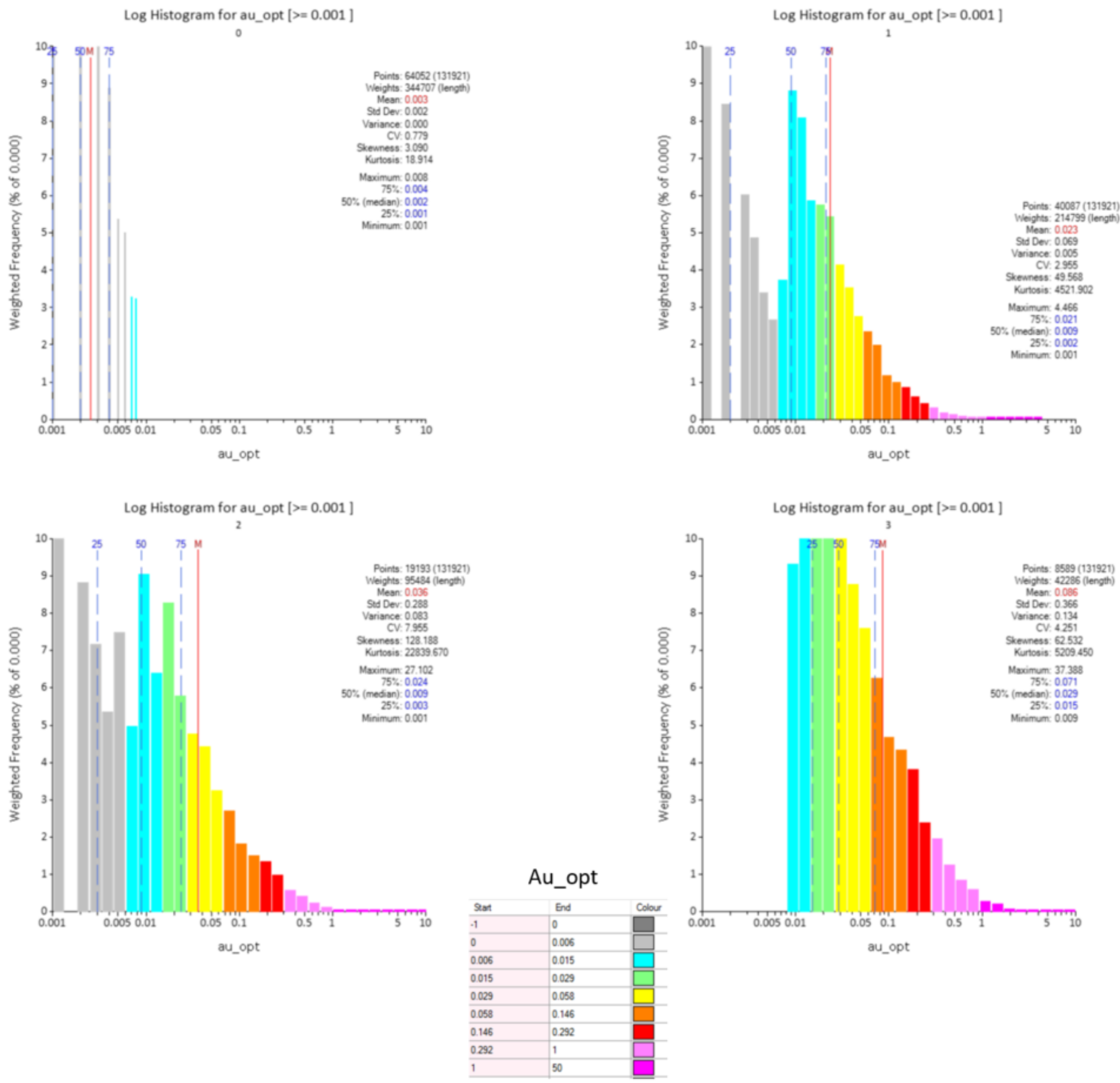
Figure 14-6: HGM16 Correlation of Au grade vs 3% Pyrite

Au-Si-Py Indicators model the spatial extent of gold, silicification, and pyrite. A series of indicators were established to evaluate the utility of a combined set of indicators based on the recognized associations.

Indicators were set up at the following thresholds:

- Au - 0.0085 oz/st (0.291 g/t)
- Si - >= 1
- Py - > 1%

Following are summary indicator tabulated values ranging from 0 to 3, based on the three individual indicator values. This summary indicator was used to establish interpolation domain shells. Gold statistics from the raw drilling data within the summary zones are presented Figure 14-7.



Source: OceanaGold

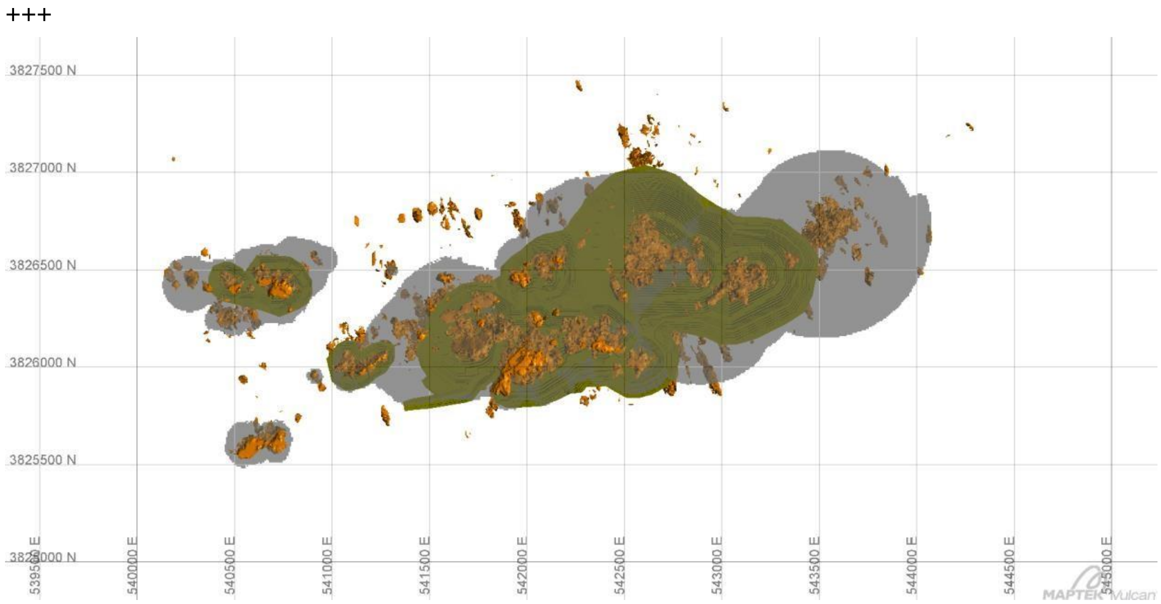
Figure 14-7: HGM16 Summary of Indicator Values ranging from 0 to 3 in oz/st

The graphs show clearly an increasing gold grade (oz/st) relationship to increasing silicification and pyrite at Haile.

14.1.6 Domains/Interpolation Shells

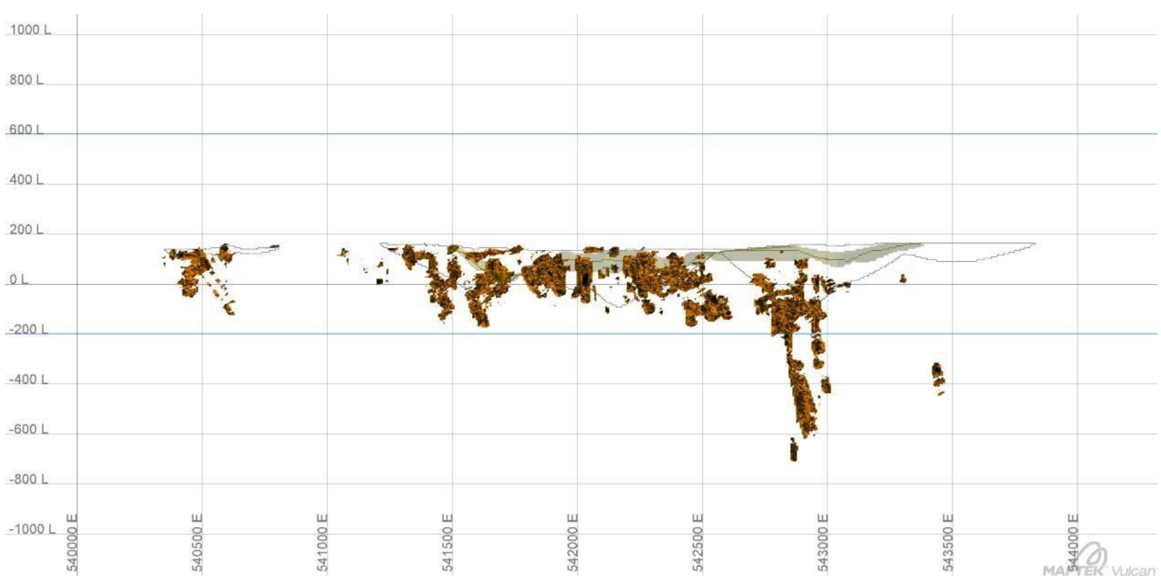
Domains/Interpolation shells – Two of the three summary indicator shells were chosen to be used in constraining gold mineralization, the outer shell is the 1/3 summary indicator shell within which at least one of the three indicators are positive.

The plan view and 3826250N long section in Figure 14-8 and Figure 14-9 show the US$1,200 IMC Resource shell in gray, IMC Reserve with modified Mill Zone design in green, and the ore interpolation Zone 1 in brown.



Source: OceanaGold

Figure 14-8: HGM16 Plan view of US$1,200 IMC Resource shell in gray vs. IMC Reserve Mill Zone in green vs ore interpolation Zone 1 in brown



Source: OceanaGold

Figure 14-9: HGM16 Long Section view of US$1,200 IMC Resource Shell in gray vs IMC Reserve Mill Zone in green vs. ore interpolation Zone 1 in brown

14.1.7 Assay Cap Values

Assay Cap Values – Prior to compositing, gold and silver grades were top-cut to reduce the impact of high-grades on the estimation. The basis for the top-cutting combined inspection of log-log cumulative distribution function plots of grade with a metal at risk analysis conducted in Excel. Top-cuts of 20.6 g/t and 72 g/t were applied to gold grades within the 1/3 summary indicator zone and the 2/3 summary indicator zone respectively.

Figure 14-10 shows the top cutting for gold in (oz/st) the 1/3 summary indicator zone. Figure 14-11 shows the top cutting for gold (oz/st) in the 2/3 and 3/3 summary indicator zone.

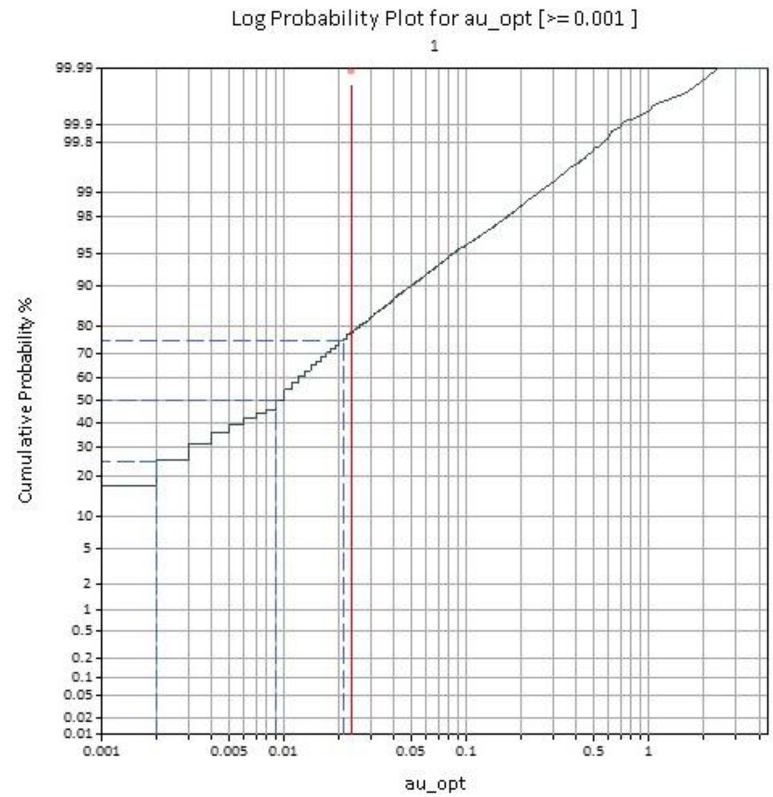
Table 14-1 summarizes the anisotropy for each zone.

Ore = 1

Top-cut = 0.6 opt

Decile	No. of Samples	Ounces gold/ton Average	Minimum	Maximum	Contained Metal Ounces	%Total
0-10	4009	0.001	0.001	0.001	20	0%
10-20	4009	0.001	0.001	0.002	24	0%
20-30	4009	0.002	0.002	0.003	45	1%
30-40	4009	0.004	0.003	0.005	75	1%
40-50	4009	0.006	0.005	0.009	130	3%
50-60	4009	0.010	0.009	0.011	219	4%
60-70	4009	0.013	0.011	0.016	299	6%
70-80	4009	0.019	0.016	0.024	447	9%
80-90	4009	0.033	0.024	0.046	767	15%
90-100	4006	0.128	0.046	4.466	2,994	60%
90-91	401	0.048	0.046	0.050	114	2%
91-92	401	0.053	0.050	0.056	124	2%
92-93	401	0.059	0.056	0.062	139	3%
93-94	401	0.066	0.062	0.070	154	3%
94-95	401	0.075	0.070	0.080	178	4%
95-96	401	0.087	0.080	0.097	209	4%
96-97	401	0.108	0.097	0.120	249	5%
97-98	401	0.139	0.120	0.160	324	6%
98-99	401	0.196	0.160	0.245	457	9%
99-100	397	0.479	0.245	4.466	1,048	21%
Total	40087	0.022	0.001	4.466	5,020	100%

0-10	4009	90-91	36482	401
10-20	8018	91-92	36883	401
20-30	12027	92-93	37284	401
30-40	16036	93-94	37685	401
40-50	20045	94-95	38086	401
50-60	24054	95-96	38487	401
60-70	28063	96-97	38888	401
70-80	32072	97-98	39289	401
80-90	36081	98-99	39690	401
90-100	4006	99-100	40087	397



Source: OceanaGold

Figure 14-10: HGM16 Top Cutting for 0.6 oz/st (20.6 g/st) within 1/3 indicator summary

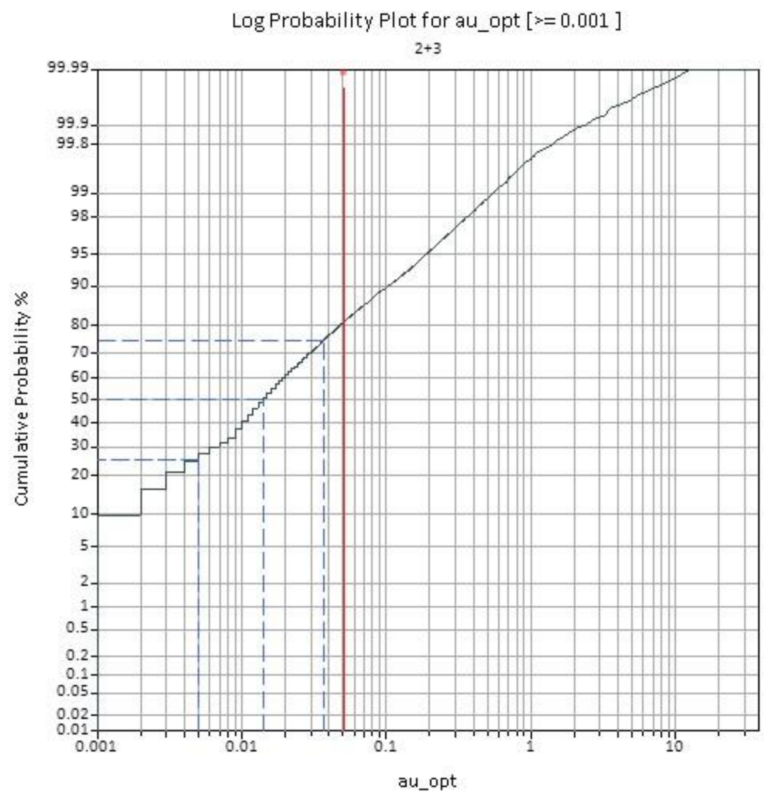
Ore = 2+3

Top-cut = 2.11 opt

	No. of	Ounces gold/ton			Contained Metal	
Decile	Samples	Average	Minimum	Maximum	Ounces	%Total
0-10	2779	0.001	0.001	0.002	14	0%
10-20	2779	0.002	0.002	0.003	33	0%
20-30	2779	0.005	0.003	0.007	64	1%
30-40	2779	0.009	0.007	0.010	120	2%
40-50	2779	0.012	0.010	0.014	166	2%
50-60	2779	0.017	0.014	0.020	231	3%
60-70	2779	0.024	0.020	0.030	338	5%
70-80	2779	0.038	0.030	0.049	537	8%
80-90	2779	0.072	0.049	0.109	997	14%
90-100	2771	0.350	0.109	37.388	4,591	65%
90-91	278	0.116	0.109	0.121	162	2%
91-92	278	0.129	0.122	0.138	181	3%
92-93	278	0.147	0.138	0.156	201	3%
93-94	278	0.156	0.157	0.177	219	3%
94-95	278	0.190	0.177	0.204	257	4%
95-96	278	0.223	0.205	0.244	299	4%
96-97	278	0.270	0.244	0.301	344	5%
97-98	278	0.343	0.303	0.402	449	6%
98-99	278	0.493	0.404	0.634	617	9%
99-100	269	1.702	0.635	37.388	1,862	26%
Total	**27782**	**0.053**	**0.001**	**37.388**	**7,091**	**100%**

0-10	2779
10-20	5558
20-30	8337
30-40	11116
40-50	13895
50-60	16674
60-70	19453
70-80	22232
80-90	25011
90-100	2771

90-91	25289	278
91-92	25567	278
92-93	25845	278
93-94	26123	278
94-95	26401	278
95-96	26679	278
96-97	26957	278
97-98	27235	278
98-99	27513	278
99-100	27782	269



Source: OceanaGold

Figure 14-11: HGM16 Top Cutting for 2.1 oz/st (72 g/t) within 2/3 indicator summary

Table 14-1: HGM16 Anisotropy

Variable	Zone	Model	Bearing	Plunge	Dip	C_0	C_1	r1Major	r1Semi-Major	r1Minor	C_2	r2Major	r2Semi-Major	r2Minor
	0	Spherical	274	-7	-19	0.1	0.34	19	6	7	0.56	95	43	94
Au	1	Spherical	250	-3	-15	0.13	0.42	6	9	12	0.45	17	45	57
	2	Spherical	271	14	32	0.07	0.36	14	16	31	0.57	32	57	64
	0	Spherical	75	1	10	0.15	0.2	99	39	3	0.14	150	64	14
Ag	1	Spherical	79	10	12	0.17	0.83	72	55	38				
	2	Spherical	303	-29	-8	0.24	0.44	36	24	10	0.32	85	66	21
S v2	All	Spherical	45	-45	83	0.3	0.45	127	150	25	0.25	176	268	100
C v2	All	Spherical	227	-7	-45	0.16	0.41	184	271	70	0.43	1110	496	439

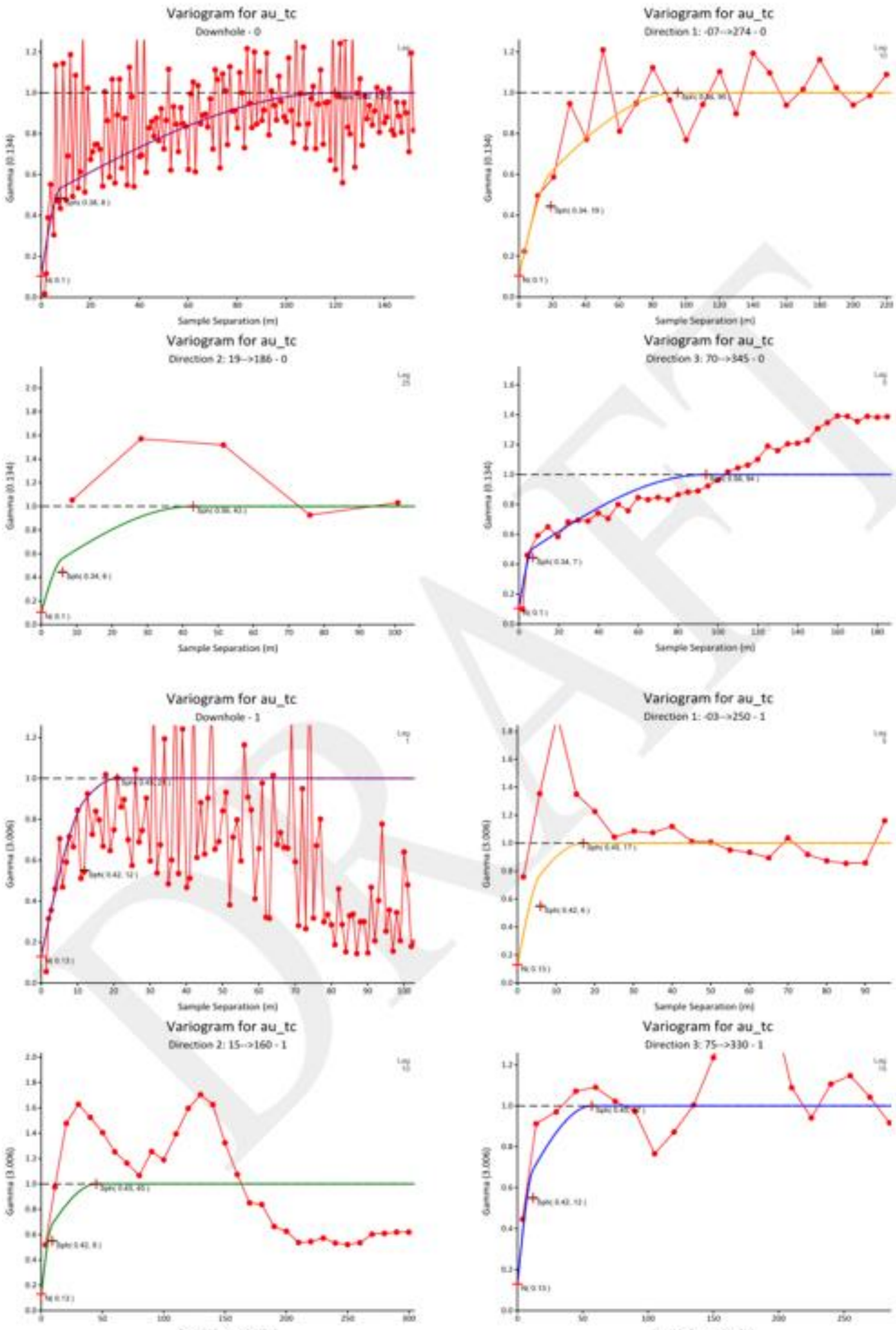
Source: OceanaGold
Zone = 0 is the 0/3 Summary Indicator Zone
Zone = 1 is the 1/3 Summary Indicator Zone
Zone = 2 is the 2/3 Summary Indicator Zone

14.1.8 Compositing

Compositing – 2.5 m bench composites for gold and silver were calculated for the HGM16 estimation. The IMC15 estimate was based upon 6.1 m bench composites. Due to the lack of data, carbon and sulfur were estimated directly from the sample grades which were collected intermittently down the hole, typically only one sample per 6.1 m of drilling.

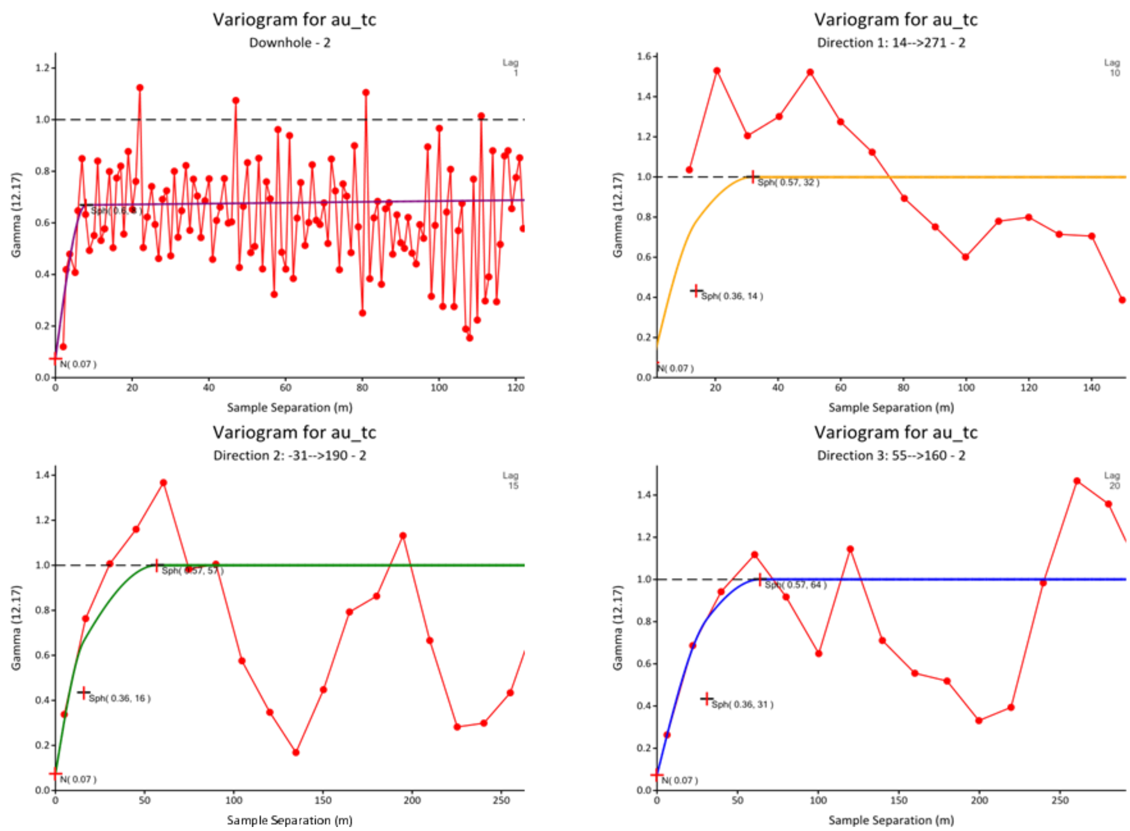
14.1.9 Variogram Analysis and Modeling

Variograms were estimated for gold and silver within and outside of the estimation domains. Variograms for carbon and sulfur were estimated globally. Figure 14-12 and Figure 14-13 show the variograms.



Source: OceanaGold

Figure 14-12: HGM16 Variograms



Source: OceanaGold

Figure 14-13: HGM16 Variograms

14.1.10 Block Model

The HGM16 resource block model was constructed in Vulcan and the parameters below in Table 14-2 are based on a Parent block size of 10 mx 10 m x 5m in x, y, z respectively and is not rotated. Within the 1/3 summary indicator shell the model was sub blocked to 5 m x 5 m x 2.5 m to better define the main ore zone and to model the mineralization at the approximate selective mining unit size in the mine. The IMC15 model did not employ sub-blocking.

Table 14-2: HGM16 Block Model Parameters

UTM NAD 83 grid	Origin Coordinates	Parent Block Sizes	Sub block Sizes
X	17.910000	10 m	5 m
Y	213.530000	10 m	5 m
Z	0.000000	5 m	2.5 m

Source: OceanaGold

14.1.11 Estimation Methodology

Gold, silver, carbon and sulfur estimation was completed using ordinary kriging.

Data was isolated for estimation by the summary indicator zones.

- Zone = 0: outside of the 1/3 summary indicator shell
- Zone = 1: between the 1/3 summary indicator shell and the 2/3 summary indicator shell
- Zone = 2: within the 2/3 summary indicator shell

Gold and silver estimation was run using four passes each with larger search distances as displayed in the figure with successively larger ellipses. Figure 14-14 perspective is oriented looking south-east, dikes are shown in red and the 1/3 summary indicator domain is displayed in the brownish color.

Source: OceanaGold

Figure 14-14: HGM16 Search Passes shown in grey ellipsoids. Search 1 smallest, Search 4 biggest

Table 14-6 summarizes the gold estimation search criteria.

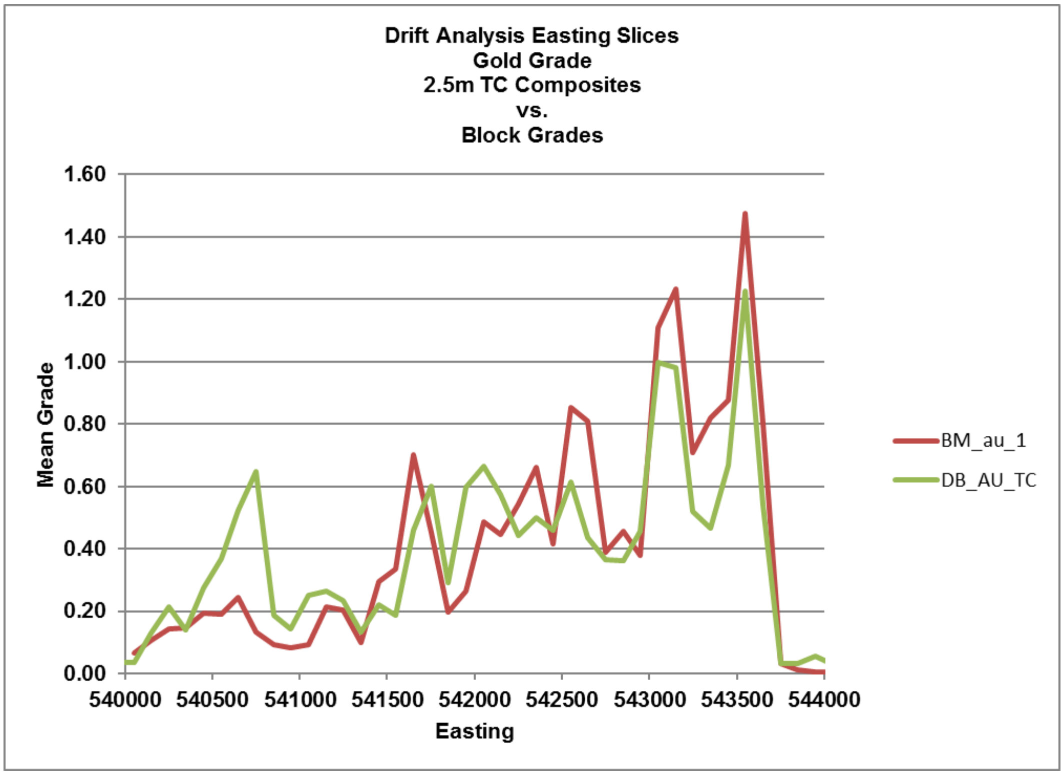
					Search Radius				Sample Counts				
Zone	**Bearing**	**Plunge**	**Dip**	**Pass**	**Major**	**Semi-Major**	**Minor**	**Vario Range Multiplier**	**Min**	**Max**	**Max per Hole**	**HY Thresh**	**Notes**
0	274	-7	-19	1	47.5	21.5	47	0.5	4	16	3	No	Top-cut of 8.3gpt applied in Vulcan
				2	95	43	94	1	4	16	3	No	
				3	190	86	188	2	1	12	3	No	
				4	285	129	282	3	1	10	3	No	
1	250	-3	-15	1	8.5	22.5	28.5	0.5	4	16	3	No	Top-cut of 20.6gpt applied
				2	17	45	57	1	4	16	3	No	
				3	34	90	114	2	1	12	3	No	
				4	136	360	456	8	1	10	3	No	
2	271	14	32	1	16	28.5	32	0.5	4	16	3	No	Top-cut of 72.3gpt applied
				2	32	57	64	1	4	16	3	No	
				3	64	114	128	2	1	12	3	No	
				4	192	342	384	6	1	10	3	No	

Source: OceanaGold

Figure 14-15: HGM16 Gold estimation search criteria

14.1.12 Model Validation

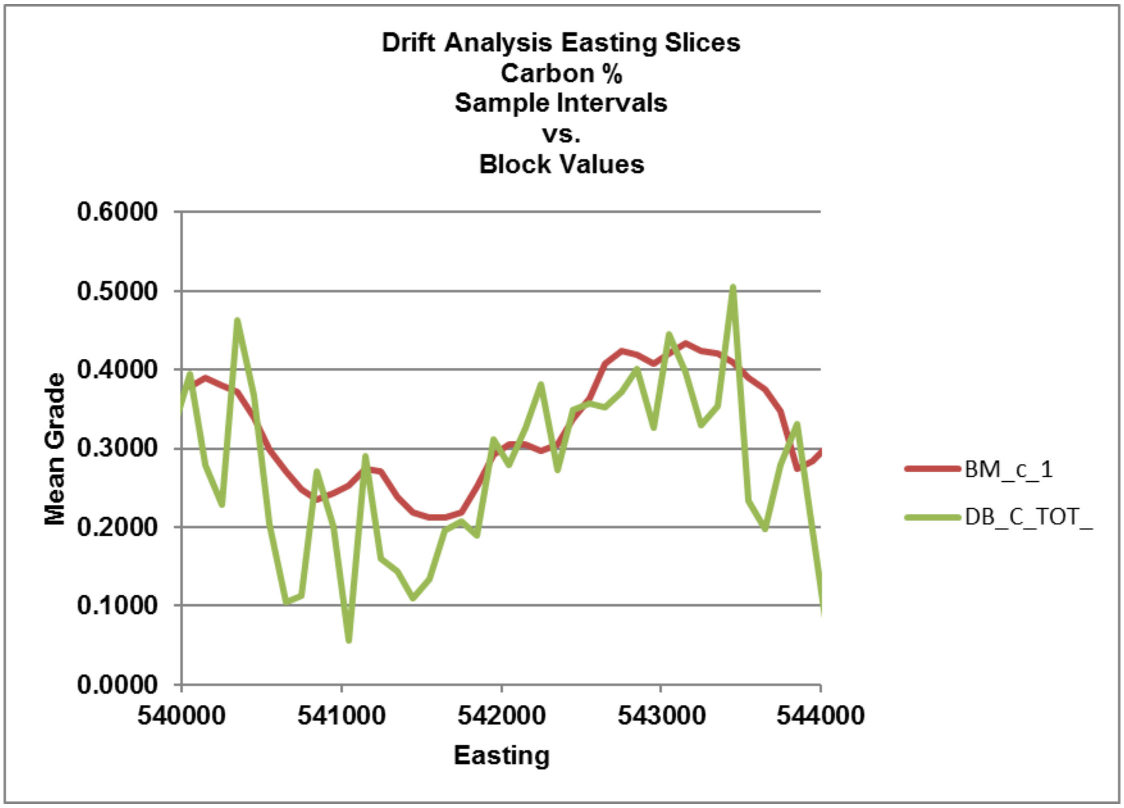
The swath plot, shown in Figure 14-17 was used to compare 2.5 m bench composite grades after top-cutting to the estimated block grades. This was used to determine the effectiveness of the estimation technique for gold and silver.



Source: OceanaGold

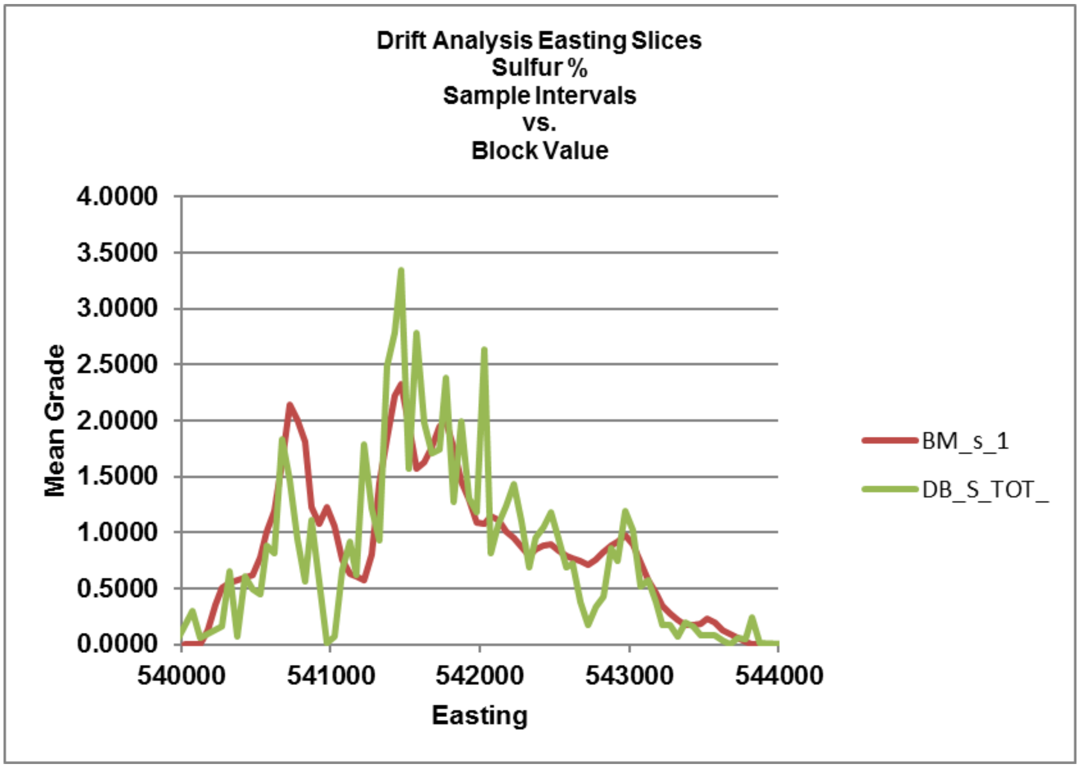
Figure 14-16: HGM16 Swath Plot 2.5m composites vs blocks with unaltered mean grade scale

Carbon and sulfur were estimated from raw drill hole data because of the lack of data relative to the gold and silver data sets. Swath plots for carbon and sulfur are shown in Figure 14-17 below.



Source: OceanaGold

Figure 14-17: HGM16 Swath Plot with Carbon vs Block Values



Source: OceanaGold

Figure 14-18: HGM16 Swath Plot with Sulfur vs Block Values

14.1.13 Resource Classification Logic

The resource classification logic applied to the HGM16 model is based on the number of drill holes included in the kriging neighborhood and the average distance to these drill holes relative to the range of the variogram used in the estimation.

- Measured classification applied to blocks if there are four or more composites found within the first pass distance. 1st pass distance is 40 m. This is 50% of the variogram range.
- Indicated classification applied to blocks if there are two or more composites found within the first or second pass distance. 2nd pass distance is 80 m. This is 100% of the variogram range.
- Inferred classification applied to blocks if there is one composite within the 1st, 2nd, or 3rd pass distance. 3rd pass distance is 160 m. This is 200% of the variogram range.
- Second pass inferred classification applied to blocks if there is one composite within the 4th pass distance. 4th pass distance is 240 m. This is 300% the variogram range.

14.1.14 Mineral Resource Statement

The component of the block model that qualifies as an open pit Mineral Resource was estimated using the Lerchs-Grossman method. The intent of the application of the Lerchs-Grossman method is to establish the component of mineralization that has reasonable prospects of economic extraction.

Table 14-3 summarizes the economic parameters used in the Lerchs-Grossman method.

Table 14-3: Lerchs-Grossman Parameters

Lerchs Grossman Parameters - HGM16 - US$1500oz shell - 0.45 g/t		
Parameter	**Unit**	**Value**
Base Mining Cost	US$/t	1.37
Incremental Mining Cost	US$/t / 5 m bench	0.00875
PAG Rehabilitation Cost	US$/t PAG waste	0.6
Underground Mining Cost	US$/t	38.69
Sustaining Capital Cost	US$/t	0
Processing Cost	US$/t ore	10.61
G&A Cost	US$/t ore	3.184
Gold Recovery	%	(1-(0.2152*au grade^-0.3696))
Silver Recovery	%	1-((0.1768*Ag+0.1877)/Ag)
Mill Throughput	Mtpa	4

Source: OceanaGold

Table 14-4 summarizes the resulting open pit resources. A US$1,500/oz shell was used with a cut-off of 0.45 g/t.

The reader is cautioned that Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the Inferred Mineral Resources will be realized or that they will convert to Mineral Reserves.

Table 14-4: Open Pit Total Mineral Resources as of January 1, 2017

Resource Summary - HGM16 model - US$1500/oz shell - 0.45 g/t			
HGM16	**Tonnes (Mt)**	**Au (g/t)**	**Au (Moz)**
Measured	7.06	1.97	0.45
Indicated	52.2	1.63	2.73
Inferred	11.1	1.4	0.49
Measured and Indicated	59.2	1.67	3.17

Source: OceanaGold

- CoG = 0.45 g/t Au and US$1500/oz.
- Tonnages are metric tonnes.
- Grades are in g/t
- Open pit resource is reported within a US$1,500/oz optimized shell.
- Mineral Resources in this table includes the Mineral Reserve.
- The open pit Mineral Resources were estimated by Bruce van Brunt, a Qualified Person

14.1.15 Mineral Resource Sensitivity

Metal price changes could materially change the estimated Mineral Resources in either a positive or negative way. The US$1,500/oz plan is seen here in Table 14-5 alongside US$1,300, US$1,700, and US$1,900 price points.

Table 14-5: Total Mineral Resources with Various Price Sensitives

		Measured		Indicated		Measured + Indicated		Inferred	
Au Price (US$/oz)	**Cut-off Au (g/t)**	**Tonnes (Mt)**	**Au (g/t)**	**Tonnes (Mt)**	**Au (g/t)**	**Tonnes (Mt)**	**Au (g/t)**	**Tonnes (Mt)**	**Au (g/t)**
1,300	0.45	7.9	1.87	49.6	1.63	57.4	1.66	8.9	1.39
1,500	0.45	7.1	1.97	52.2	1.63	59.2	1.67	11.1	1.4
1,700	0.35	8.9	1.70	65.8	1.31	74.6	1.35	12.7	1.07
1,900	0.32	9.2	1.66	72.2	1.20	81.4	1.25	14.7	0.94

Source: OceanaGold

14.1.16 Relevant Factors

At this time, there are no unique situations in relation to environmental, socio-economic or other relevant conditions that would put the Haile Mineral Resource at a higher level of risk than any other developing resource within the United States, or that would materially affect the Mineral Resource estimates.

14.2 Underground Mineral Resource Estimate

The mineralization at the Horseshoe deposit is valued only for its gold content and only gold grades were estimated in the work described in this report. Dr. Bart Stryhas constructed the geologic and Mineral Resource model discussed below. He is responsible for the resource estimation methodology, Mineral Resource classification and resource statement. Dr. Stryhas is independent of the OceanaGold applying all of the tests in Section 1.5 of NI 43-101.

The resource estimation is based on the current drill hole database, interpreted lithologies, geologic controls and current topographic data. The resource estimation is supported by drilling and sampling current to November 1, 2016. The estimation of Mineral Resources was completed utilizing a resource block model constructed using Vulcan™ modeling software.

14.2.1 General Geology and Geologic Model

The Horseshoe deposit is the highest grade and easternmost known gold deposit in the Haile district. It is a Neoproterozoic gold deposit with strong silification and pyrite overprinted by intense ductile deformation and shearing. The deposit is one of several located near the contact between metamorphosed volcaniclastic and sedimentary rocks within the upper Persimmon Fork Formation. These units have been deformed by high strain, isoclinal folding and ductile shearing which has developed a pervasive foliation striking east-northeast and dipping moderately to the northwest.

The mineralization is hosted within tightly to isoclinally folded metasediments within an open anticline trending east-northeast and dipping gently north-northwest. All units are cut by post mineralization diabase dikes striking northwest with near-vertical dips, notably along the western margin of the Horseshoe deposit.

SRK has constructed a geologic model which includes the three bedrock lithologies mentioned above as well as an un-mineralized saprolite overburden. These four rock types constitute the lithologies coded in the block model. This has resulted in a detailed, 3D geologic model created by Leapfrog.

14.2.2 Geologic Model and Controls on Gold Mineralization

SRK constructed a 3D geologic model using Leapfrog® software that includes three bedrock lithologies (metasediments, volcanics, diabase) and saprolite. These four rock types constitute the lithologies coded in the block model.

Gold mineralization is controlled mainly by structural fabric and lithology. The mineralization is concentrated in two main zones, which connected together, form a “horseshoe” geometry. The lower zone is clearly controlled by lithology where the gold mineralization is localized in deformed vein style zones of silicification within the metasediment parallel to the metavolcanic contact. This zone strikes NE-SW and dips verticay. The upper zone appears to be localized in planar zones along foliation and also in deformed vein style zones of silicification. This zone also strikes NE-SW but dips at 40º to 45º

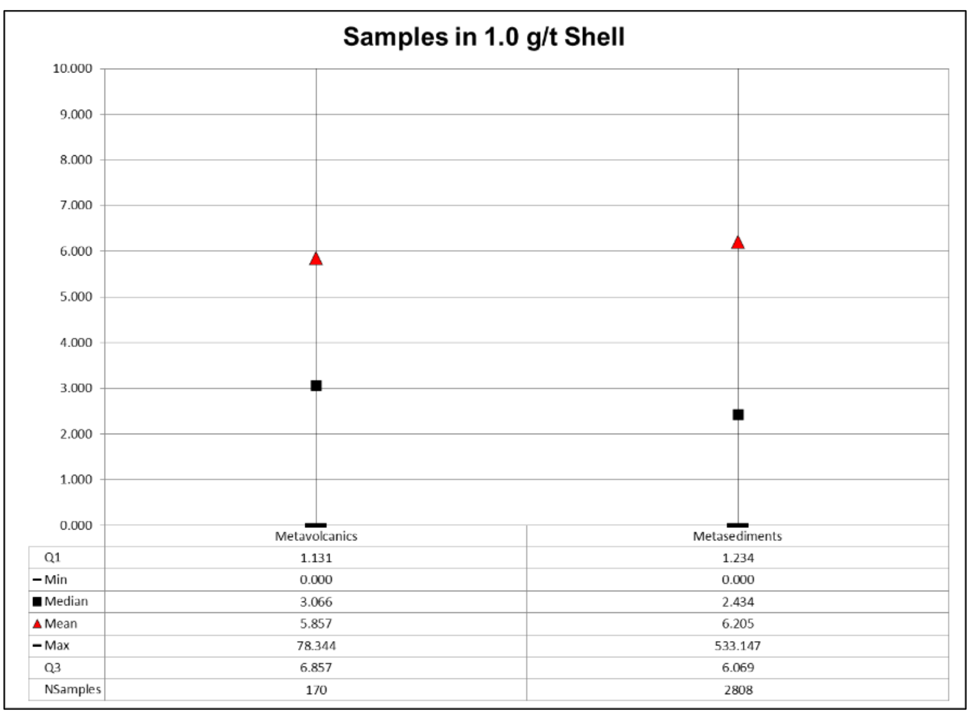
NW. OceanaGold has completed a program of oriented core drilling which has provided valuable information to better understand the structural geology of the deposit. All structural orientation data to date was acquired and plotted on lower hemisphere stereonets. The structural fabric data includes; foliation, shear planes, lithologic contacts and veins. The results of the stereonet plots are summarized in Table 14-6.

Table 14-6: Average Orientations of Structural Fabrics

Fabric	Strike	Dip (°)	Measurements (No.)
Foliation	N62E	-65NW	462
Bedding	N66E	-65NW	53
Lith. Contacts	N55E	-65NW	45
Veins	N55E	-70NW	103

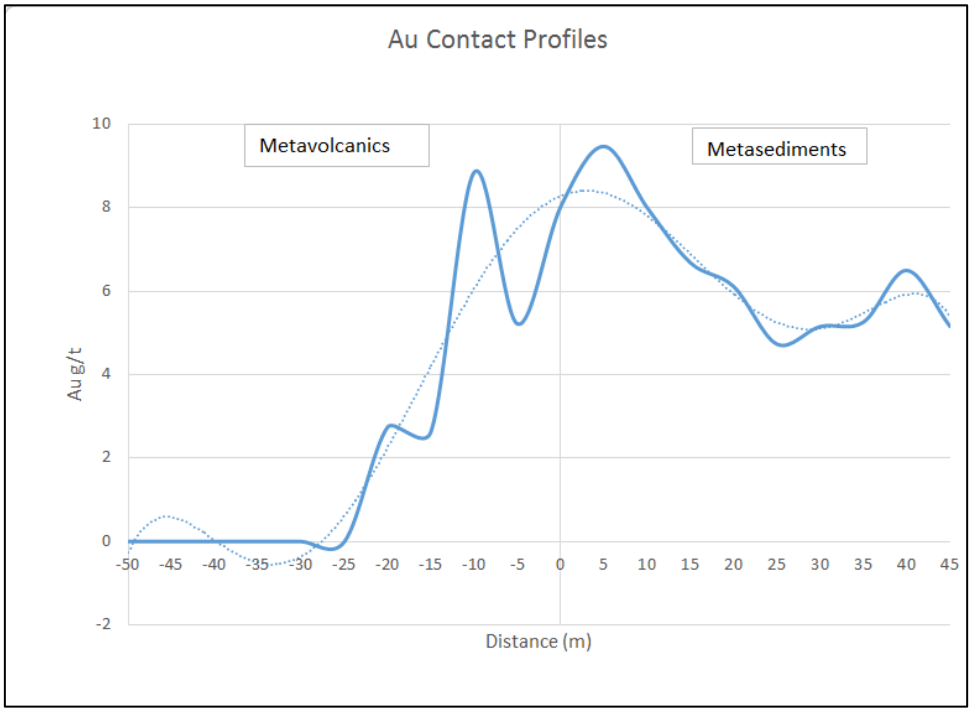
Source: SRK, 2015

To check the importance of lithologic control of mineralization, SRK constructed a box plot of gold values hosted within a 1.0 g/t Au grade shell subdivided by lithology. The results are presented in Figure 14-19. SRK also constructed a contact profile analysis to evaluate the distribution of gold along these contacts (Figure 14-20). The box plot shows the majority of mineralization is hosted in the metasediment however; there is little difference in the relative gold concentration between the two units. The contact profile shows that the majority of mineralization within the metavolcanics is located proximal to its contact with the metasediment. Only these two lithologic units were estimated for gold content and there was no hard boundary used between them.



Source: SRK, 2016

Figure 14-19: Box Plot of Gold Grade by Lithology



Source: SRK, 2016

Figure 14-20: Contact Profile of Gold Grade by Lithology

14.2.3 Density

Density testing was performed on drill core during 2016 with, 3,607 density measurements collected from all rock types. The averages of each lithology are listed in Table 14-7. Densities were assigned in the block model based on the lithology of the block.

Table 14-7: Densities Assigned in the Block Model

Rock Type	Number of Measurements	Minimum (g/cm^3)	Maximum (g/cm^3)	Average Density (g/cm^3)
Saprolite	49	1.74	4.44	2.45
Diabase Dikes	123	2.25	3.92	2.88
Metavolcanics	1,916	1.03	5.63	2.75
Metasediments	3,388	1.02	5.71	2.82
Total	**5,476**			

Source: SRK, 2016

14.2.4 Sample Database

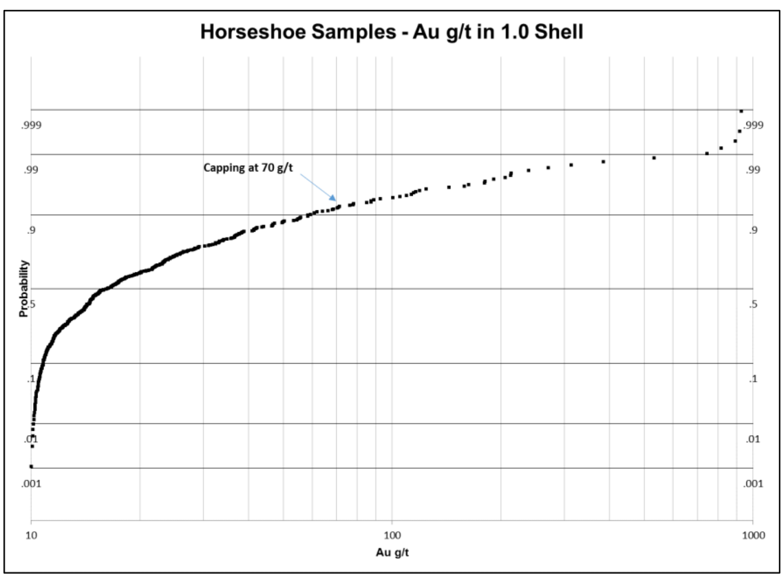
The November 1, 2016 database contains information from 4,314 diamond core and RC drill holes. The drilling was completed in two main campaigns. Three previous owners drilled the majority of the holes over the entire area of mineralization between 1981 and 2015. OceanaGold completed a focused diamond core drilling program at Horseshoe during 2015 and 2016, this included 65 holes.

The database includes four Excel files containing information on collar locations, downhole surveys, lithology and gold assays. There are 363,095 valid entries in the assay file with an average sample length of 1.6 m.

14.2.5 Capping and Compositing

The original drill hole gold values were assessed for statistical outliers using a lognormal cumulative distribution plot and decile analysis. The decile analysis was used to identify the appropriate bin range for capping and the cumulative distribution plot was used to define the final capping level. The results of the cumulative distribution plot are presented in Figure 14-21. The Au capping level was chosen at 70 g/t mainly because this is the point where the cumulative distribution trends lose continuity, and the data values above show irregular distribution. Additionally, the percentage of metal lost, impact on coefficient of variance (CV) and validation sensitivity were evaluated to derive this capping level. The Au capping was performed in a two-stage process. First, all samples were composted to 1 m to remove the impact of any short sample intervals. Next, the samples were capped to 70 g/t and then composited to 3 m run lengths. This resulted in 32 samples ranging from 72 to 929 g/t being reduced to 70 g/t prior to compositing. This was a net loss of 12% of all gold in the database. Table 14-8 lists the results of the capping and compositing.

Compositing was completed in 3-m downhole lengths with no breaks at lithologic contacts. The 3-m length was chosen as an appropriate size for two reasons. This length includes two capped assay intervals so that it provides some smoothing of the data while still preserving the variability of the gold mineralization. The 3-m composite length also results in approximately two composites being included within the diagonal intersection of the 5 m, X, Y, Z direction block size.



Source: SRK, 2016

Figure 14-21: Log Normal Cumulative Distribution Plot of Gold Assays above 10 g/t

Table 14-8: Capping and Compositing Results

Data	Zone	Mean Au (g/t)	Maximum Au (g/t)	% Change Total Au Metal	CV	Number of Samples
Original	Upper	10.2	920		4.4	1,453
Original	Lower	7.1	746		4.2	1,278
1 m No Cap	Upper	7.8	345	-0.1	2.7	1,722
1 m No Cap	Lower	6.3	352	0.6	2.3	1,686
3 m Capped 70 g/t	Upper	5.8	70	-11.7	1.5	679
3 m Capped 70 g/t	Lower	5.3	70	-0.2	1.2	666

Source: SRK, 2016

14.2.6 Block Model

The block model limits of the SRK resource estimations are listed below. The block model coordinates are referenced to the UTM NAD83 coordinate system. The block dimensions are based on a compromise between the average drill hole spacing, a typical underground stope selective mining unit and the variability of the mineralization. The block model is rotated 60° clockwise in order for the block edges to align parallel to the local foliation fabric and the stope designs. The origin, limits and block sizes are listed in Table 14-9. There are 7,086,240 blocks in the model.

Table 14-9: Block Model Size and Extents

Orientation	Origin (m)	Extent (m)	Block Dimension (m)
Easting	543,100	625	5
Northing	3,827,000	700	5
Elevation (AMSL)	-350	525	5

Source: SRK, 2016

14.2.7 Estimation Strategy

SRK constructed Leapfrog® software generated wireframe solids which enclose anomalous gold mineralization at a 1.25 g/t Au threshold. The grade shells were constructed using interpreted trend planes of mineralization. The trend planes were developed by digitizing section profiles of gold continuity which were then triangulated into 3D planes of gold continuity. The upper zone of mineralization utilized two trend planes which essentially represent the hanging wall and footwall of mineralization. The lower zone utilized two additional trend planes. Numerous grade shells were constructed using a variety of sensitivities. The grade shells were evaluated for validity and functionality using three methods. First, they were queried to determine what percentage of the available 3 m composite samples above 1.25 g/t Au are captured. The final grade shell captured 93% of all 3 m composite samples with Au grade above 1.25 g/t. Second, it was queried to determine how many samples within the grade shell were above the 1.25 g/t threshold. In the final grade shell, 98% of the samples were above the threshold. Therefore, it has a 2% dilution of the composited 3 m samples. Third, it was evaluated to access the internal dilution of the non-composited original samples. This showed that the grade shell has 78% of original samples above the 1.25 threshold and therefore has an actual dilution of 22%. Fourth, the final grade shell was visually inspected to be sure the geometry was reasonable, based on the nearby drill holes. The visual review resulted in removal of two small satellite pods of mineralization which were distal to Horseshoe and more proximal to the nearby Palomino anomalous gold zone.

As described above, the gold mineralization is hosted in two domains each with a unique orientation. For each zone, the trend planes used to guide the grade shells construction were translated outward to capture all model blocks enclosed by the grades shell. These translated planes were then used to guide a dynamic search orientation utilized in the gold grade estimation. Both OK and Inverse Distance Weighting Squared (IDW^2) algorithms were used for the grade estimations to evaluate the deposits sensitivity to estimation algorithms.

14.2.8 Estimations Procedures

The grade estimations utilize a three-pass sample search strategy with each pass searching longer distances than the previous. The search distances and variogram parameters are listed in Table 14-10 and Table 14-11, respectively. For all estimations, the following criteria were used:

- Dynamic search orientation essentially parallel to the plane of gold continuity for each zone;
- Minimum of three composites and maximum of eight composites to estimate grade;
- Sample length weighting to account for any short composites located at the ends of drill holes;
- Composites from a minimum of two drill holes; and
- Composites from a minimum of two octants.

As part of the grade estimation, model validation is conducted as an interactive process. To achieve proper validation, no higher-grade composites were limited by the distance they could be interpolated. Figure 14-22 and Figure 14-23 show representative cross sections of the gold estimation results.

Table 14-10: Au Grade Estimation Search Distances

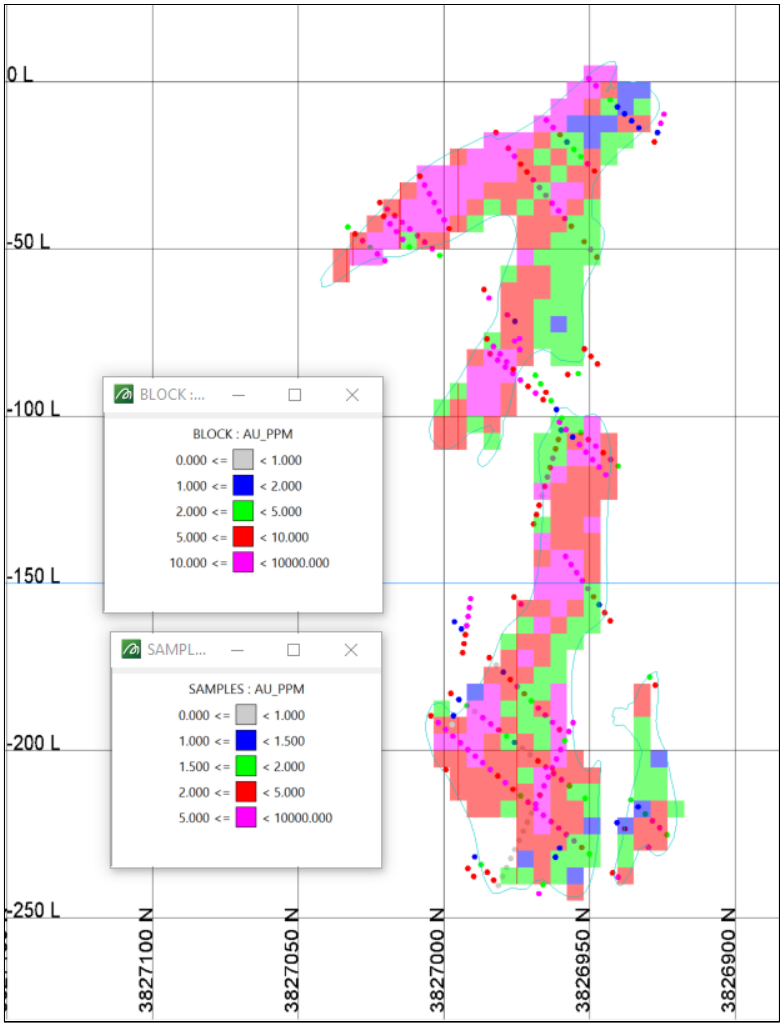
Estimation	Estimation Pass	Search Range (m) (X, Y, Z)
All	1	15,15,5
	2	25,25,10
	3	75,75,15

Source: SRK, 2016

Table 14-11: Ordinary Kriging Parameters

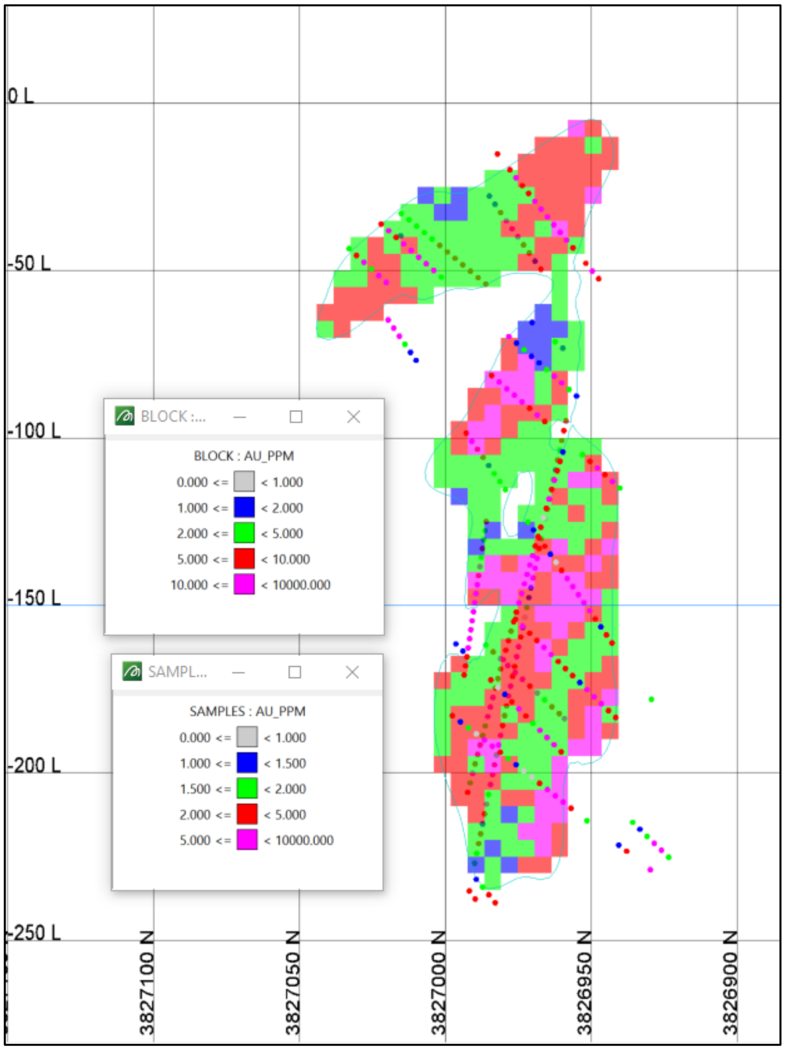
Estimation Domain	Variogram Structure	Nugget	Sill Differential	Rotations (Vulcan, X, Y, Z)	Ranges (m) (, X, Y, Z)
Upper	1st Exponential	0.3	0.40	240°,-5°,-35°	7, 7, 12
	2nd Spherical		0.15		7, 80, 12
	3rd Spherical		0.15		50, 80, 12
Lower	1st Exponential	0.35	0.45	Isotropic	10, 10, 10
	2nd Spherical		0.40		35, 35, 35

Source: SRK, 2016



Source: SRK, 2016

Figure 14-22: Representative Cross Section with Estimated Au Grades (Viewing N60°E)



Source: SRK, 2016

Figure 14-23: Representative Cross Section with Estimated Au Grades (Viewing N60°E)

14.2.9 Model Validation

Six techniques were used to evaluate the validity of the block model. First, the interpolated block grades were visually checked on sections, plan views and in 3D for comparison to the composite assay grades. Second, the general model estimation parameters were reviewed to evaluate the performance of the model with respect to supporting data. This included the number of composites used, number of drill holes used, average distance to samples used, and the number of blocks estimated in each pass. The results of this analysis are presented in Table 14-12. These show that the blocks are well-informed from sufficient samples, and are selected from multiple drill holes at a reasonable distance. Third, statistical analyses were conducted, comparing the estimated block grades from the OK and IDW^2 estimations to the composite sample data supporting the estimation. Table 14-13 lists the results of the statistical comparison. In all cases, the average block grades of the upper zone are slightly below the composite grades as desired. In the lower zone, the block grades are slightly higher than the composites because the estimation utilizes samples from both zones near its upper part and the sample grades in the upper zone are slightly higher than those in the lower zone, thus skewing the averages. When evaluating both zones combined, both of the estimation algorithms produce block grades very close to the composite grades. Fourth, a nearest neighbor (NN) estimation was run using a single composite to estimate each block using the same parameters as the OK and IDW^2 estimations. The total contained metal, at a zero CoG in the nearest neighbor estimation, is compared to the OK and IDW^2 estimation at the same cut-off. The results of this comparison are listed in Table 14-14. The nearest neighbor estimation shows slightly more metal than the OK estimation as desired and nearly identical metal to the IDW^2 estimation. The fifth validation was to construct multiple oriented swath plots located with relatively equal representation around the curvature of the horseshoe geometry. Figure 14-24 shows the location of the swath plots. The results of the analysis are shown in Figure 14-25, which illustrate good correlation between block and composites grades with an appropriate amount of smoothing. The final validation was an assessment of the impacts of edge dilution about the margins of the Au grade shell. To quantify the impacts of dilution, a partial Au grade estimation was completed for all blocks touching the grade shell. These blocks were estimated with samples internal to the wireframe and then again with the samples external to the wireframe and a final diluted Au grade was calculated based on each of the grade estimations weighted by the proportion of the block representing the estimation. The results of the diluted model are compared to the undiluted model for all blocks with 50% or greater volume internal to the grade shell in Table 14-15. This shows there is an approximate 10% net loss in contained metal in the diluted model related to both decrease in grade and tonnes.

Based on the results of the various model validations, the OK estimate was chosen as the final Mineral Resource estimation. All resource reporting tables are based on this estimation.

Table 14-12: Estimation Performance Parameters of Au Estimation in Grade Shell

Domain	Estimation	Samples Used (Number)	Drill holes Used (Number)	Average Distance to Samples (m)	Blocks Estimated (%)
Upper	Pass 1	4	2.3	9	37
	Pass 2	6	3.1	15	52
	Pass 3	7	4.3	29	11
	All Passes	**5**	**3.0**	**14**	**100**
Lower	Pass 1	4	2.3	8	29
	Pass 2	5	2.9	15	50
	Pass 3	7	4.0	29	21
	All Passes	**5**	**3.0**	**16**	**100**

Source: SRK, 2016

Table 14-13: Model Validation Statistical Results in Grade Shell

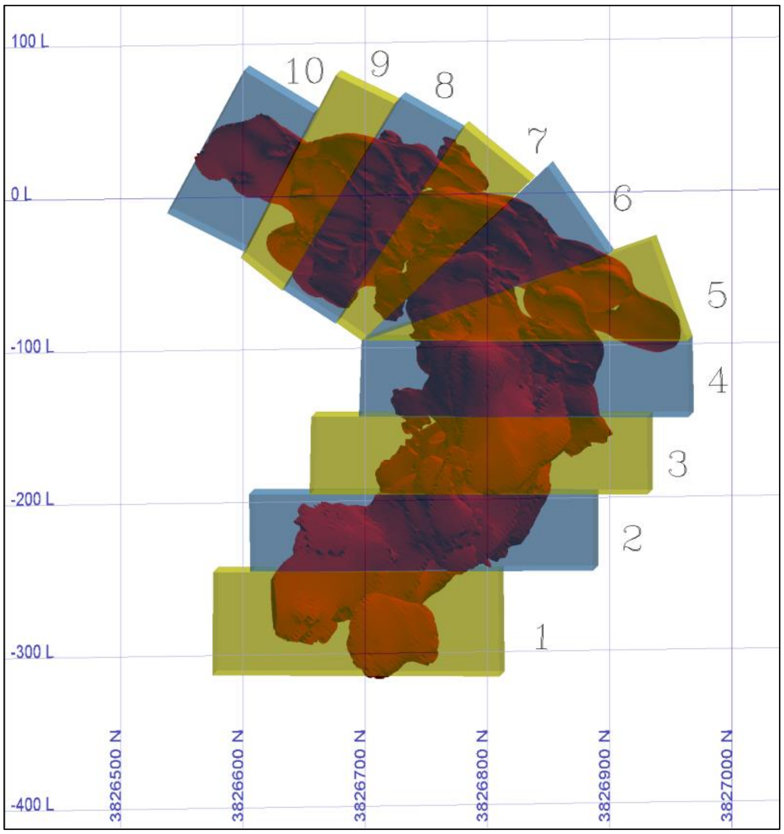
Estimation	Average Capped Composite Grade (g/t)	Average Block Grade (g/t)	Difference of Composites to Blocks (%)
Upper Zone OK	6.68	6.54	2.1
Lower Zone OK	5.84	6.27	-7.3
All Zones OK	**6.27**	**6.41**	**-2.2**
Upper Zone IDW^2	6.68	6.59	1.5
Lower Zone IDW^2	5.84	6.50	-11.4
All Zones IDW^2	**6.27**	**6.55**	**-4.4**

Source: SRK, 2016

Table 14-14: Model Validation nearest Neighbor Results in Grade Shell

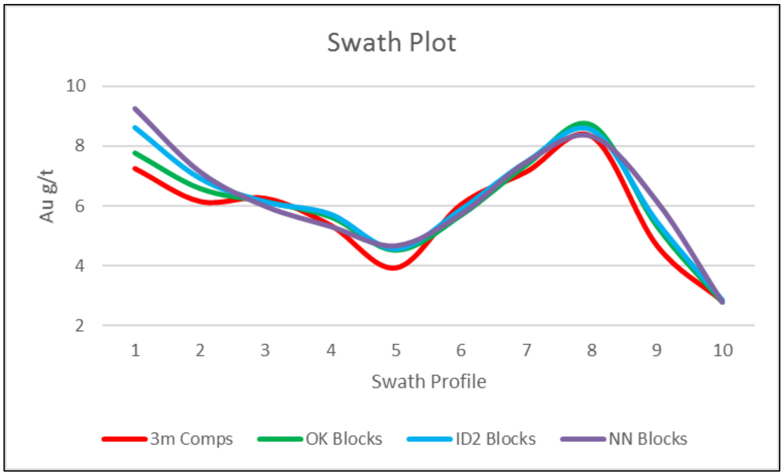
Estimation	Cut-off (g/t)	Tonnes (Mt)	Estimation Grade (g/t)	NN Au Grade (g/t)	% Difference of Metal Mass, NN to Estimated
Upper Zone OK	0	1.69	6.55	6.62	1.1
Lower Zone OK	0	1.65	6.26	6.57	4.6
All Zones OK	**0**	**3.35**	**6.41**	**6.59**	**2.8**
Upper Zone IDW^2	0	1.70	6.59	6.62	0.5
Lower Zone IDW^2	0	1.65	6.51	6.57	0.9
All Zones IDW^2	**0**	**3.14**	**6.55**	**6.36**	**0.7**

Source: SRK, 2016



Source: SRK, 2016

Figure 14-24: Variously Oriented Swath Domains (Viewing Northwest)



Source: SRK, 2016

Figure 14-25: Swath Plot Results

Table 14-15: Impacts of Dilution

Cut-off	Undiluted OK			Diluted OK			Percentage Difference Diluted to Undiluted		
Au (g/t)	Au Grade (g/t)	Tonnes (Mt)	Au Ounces (koz)	Au Grade (g/t)	Tonnes (Mt)	Au Ounces (koz)	Au Grade (g/t)	Tonnes (Mt)	Au Ounces (koz)
1.00	6.23	3.35	690.4	5.68	3.35	649.6	-10	0	-6
1.25	6.23	3.35	690.5	5.72	3.33	649.0	-9	-1	-6
1.50	6.25	3.33	689.3	5.84	3.27	646.1	-7	-2	-7
1.75	6.36	3.25	685.3	6.04	3.15	640.1	-5	-3	-7
2.00	6.50	3.17	680.2	6.24	3.03	632.6	-4	-5	-8

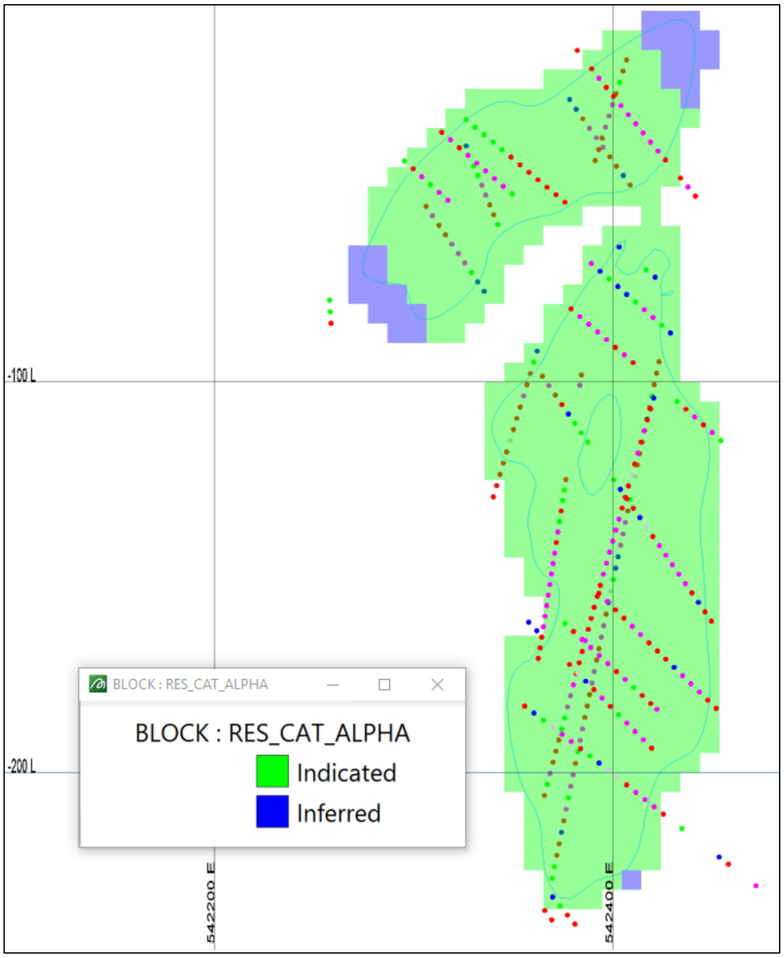
Source: SRK 2016

14.2.10 Resource Classification

Mineral Resources are classified under the categories of Measured, Indicated and Inferred according to CIM guidelines. Classification of the Mineral Resources reflects the relative confidence of the grade estimates and the continuity of the mineralization. This classification is based on several factors including sample spacing relative to geological and geo-statistical observations regarding the continuity of mineralization, data verification to original sources, specific gravity determinations, accuracy of drill collar locations, accuracy of topographic data, quality of the assay data and many other factors which influence the confidence of the mineral resource estimation. No single factor controls the Mineral Resource classification, rather each factor influences the end result.

The Mineral Resources reported for the Horseshoe deposit are classified as Indicated and Inferred Mineral Resources, based primarily on drill hole spacing since all other supporting data is of good quality. A wire frame solid was constructed around the areas where the majority of the blocks were estimated in the first or second pass of the estimation which had an average search distance of less than 10 m and 15 m respectively. These wireframe solids were used to assign the Indicated Mineral Resource classification. All blocks outside of the Indicated wireframes were classified as Inferred

Mineral Resources. Figure 14-26 shows a representative cross section of the final resource classification.



Source: SRK, 2016

Figure 14-26: Representative Cross Section Showing Resource Classification (Viewing N60°E)

14.2.11 Final Diluted Resource and Reserve Block Model Estimate

The block model supporting the final Mineral Resource and reserve was constructed and estimated in the identical manor as the initial Mineral Resource model with the following exceptions. The grade of each block, which intercepted the 1.25 g/t Au grade shell was assigned dilution. This was achieved by estimating those blocks twice; first using the samples internal to the grade shell and second using the samples external to the grade shell. The final grade of these blocks are calculated using the weighted proportions of the block internal and external to the grade shell against the respective estimated grades. These procedures generate a diluted resource model, which is used to support the final resource tables and all reserve stope designs and scheduling.

14.2.12 Mineral Resource Statement

The Horseshoe Mineral Resource statement is based on the OK model as presented in Table 14-16. The resource is not confined within any conceptual stope designs and a CoG of 1.17 g/t Au has been applied. The CoG assumes underground mining methods and is based on a mining cost of US$35/t, milling cost of US$10/t, administration cost of US$5.60/t, a gold price of US$1,500/oz, and a gold recovery of 90%.

Table 14-16: Horseshoe Underground Mineral Resource Statement as of November 1, 2016 SRK Consulting (U.S.), Inc.

Classification	Au Cut-Off (g/t)	Tonnes (Mt)	Au (g/t)	Contained Au (Moz)
Indicated	1.17	2.7	5.68	0.50
M & I	1.17	2.7	5.68	0.50
Inferred	1.17	1.2	5.0	0.20

Source: SRK, 2016

- Mineral Resources are not ore reserves and do not have demonstrated economic viability.
- All figures rounded to reflect the relative accuracy of the estimates.
- Metal assays were capped where appropriate.
- CoG is based on a mining cost of US$35/t, milling cost of US$10/t, administration cost of US$5.60/t, a gold price of US$1,500/oz. and gold recovery of 90%.
- The underground Mineral Resources were estimated by Bart Stryhas, a Qualified Person.

14.2.13 Mineral Resource Sensitivity

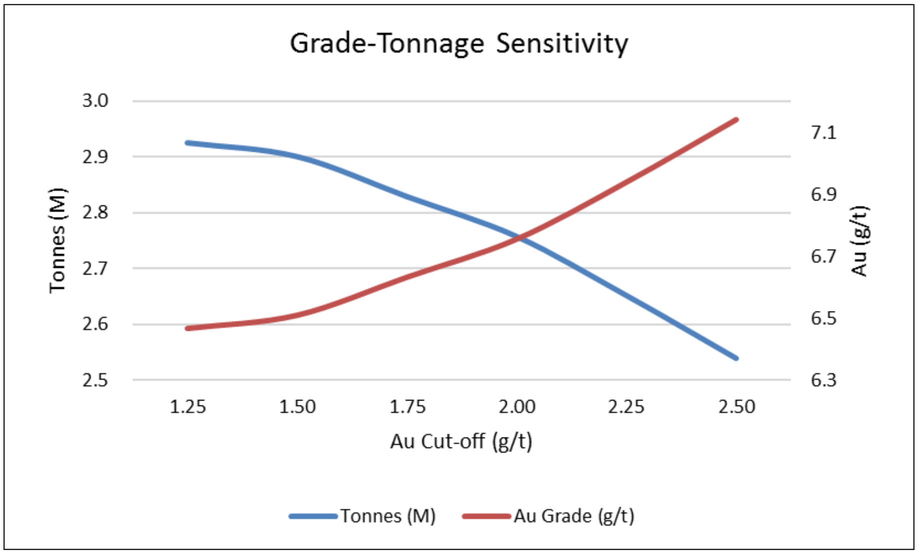
The Mineral Resources shown in Table 14-17 are presented at a range of CoGs, subdivided by resource classification. Graphical representations of the grade and tonnage sensitivities of the Measured and Indicated resources are presented in Figure 14-27. Resources are not confined within any conceptual stope design.

Table 14-17: Mineral Resource Sensitivity (1)

Indicated			
Cut-off	**Au (g/t)**	**Tonnes (Mt)**	**Au (koz)**
1	5.49	2.83	500
1.17 (2)	5.68	2.71	496
1.5	5.98	2.54	488
1.75	6.22	2.41	482
2	6.46	2.28	474

Source: SRK, 2016

(1) Tonnes and grade have been rounded to reflect the level of expected accuracy.
(2) Base Case CoG.



Source: SRK, 2016

Figure 14-27: Sensitivity of Indicated Resource Tonnes and Grade to Cut-off

14.2.14 AP-NP Block Model Estimate

A block model estimation was also created to evaluate the Acid Generating Potential (AP) and Acid Neutralization Potential (NP) of the proposed mining plan.

Database and Block Model

The database supporting the AP-NP block model includes 78 static test samples. The test work is described in detail in SRK's March 2017 report "Geochemical Summary Report, Feasibility Study, Haile Gold Mine, Lancaster County, South Carolina". The test work essentially includes the AP and Acid Neutralization Potential (NP) values for a variety of lithologic types located throughout the potential underground mining area. Table 14-18 lists the number of static test samples by lithology and basic characteristics of the data.

The reserve block model was used to generate the AP-NP model since it is used primarily in the mining plan. Details of the block model are presented in Section 1.7. Each block is 5 m by 5 m by 5 m in the X, Y, and Z directions respectively. Each model block is assigned one of four lithologic types according to the geologic model supporting the resource estimation.

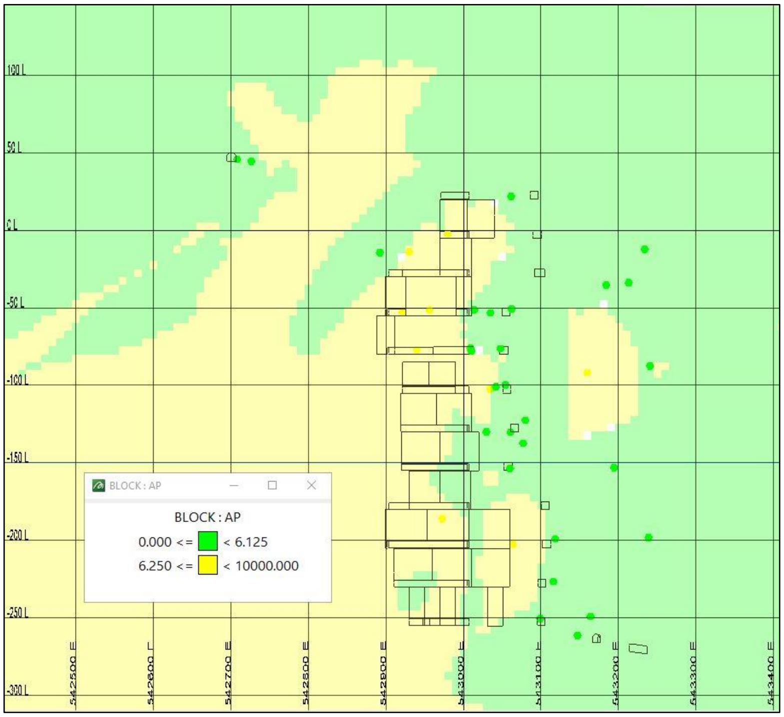
Estimation Procedures

The spatial distribution of the NPR, NNP and S were estimated for the dominant waste rock lithologies (metasediments and metavolcanics). Each of the two lithological units are considered as independent domains for the assignment and estimation of their AP and NP values. The diabase dikes were not estimated due to their benign geochemical signatures (all samples "green") and relatively minor contribution to the anticipated waste rock stream. For each lithology, the average AP and NP and S values from the analytical results (Table 14-18) were first assigned in the block model to establish background values. The actual estimation of NP and S was then conducted over the background values. The database supporting the estimation included all data values. The estimation was completed using Vulcan software and utilized a three-pass search strategy with each sequential pass searching longer distances. The search ellipsoids are oriented based on the fabric of the lithologies similar to the gold grade estimation, as this fabric controls the distribution of sulfides influencing the NP and S_T. An upper zone is defined where the fabric is moderately inclined to the NW and a lower zone is defined with a vertical fabric orientation. The search ranges are set to ensure the estimation finds data values from multiple samples while not extrapolating unreasonable distances outward from unconfined samples. The general estimation parameters are listed in Table 14-19. An inverse distance squared (IDW^2) algorithm was used to estimate grade. Representative cross sections of the AP/S_T, NP and NNP estimation are presented in Figure 14-28, Figure 14-29 and Figure 14-30, respectively. Once the final AP/S_T and NP were estimated, the neutralization potential ratio (NPR) was calculated as NPR=NP/AP and the net neutralization potential (NNP) was calculated at NNP = NP – AP. The AP shell represents the S_T aspect of the waste rock classification. The models were then used to estimate the volume of waste rock that would be generated by construction of the infrastructure, based on the Haile waste rock classification scheme.

Model Validation

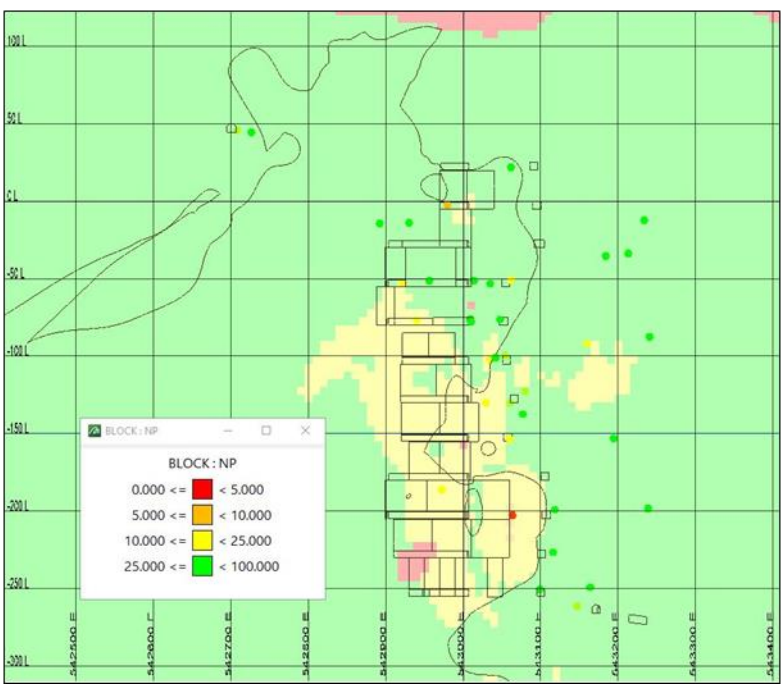
Three techniques were used to evaluate the validity of the block model. First, the interpolated block grades were visually checked on sections, plan views and in 3D for comparison to the support samples. Second, the general model estimation parameters were reviewed to evaluate the performance of the model with respect to the supporting data. This included the number of composites used, the average distances between samples and the number of blocks estimated in each pass. The results of this analysis are presented in Table 14-20. These illustrate that the estimated blocks are reasonably well informed from sufficient samples. Third, statistical analyses were made comparing the estimated block values from the IDW^2 estimation to the composite sample data supporting the estimation. Table 14-21 lists the results of the statistical comparison. In most cases, the block values are reasonably close to the samples.

The resulting model is a reasonable prediction of the AP/S_T and NP based on the relative sampling density. These results are used to partially support the geochemical model of the mining plan.



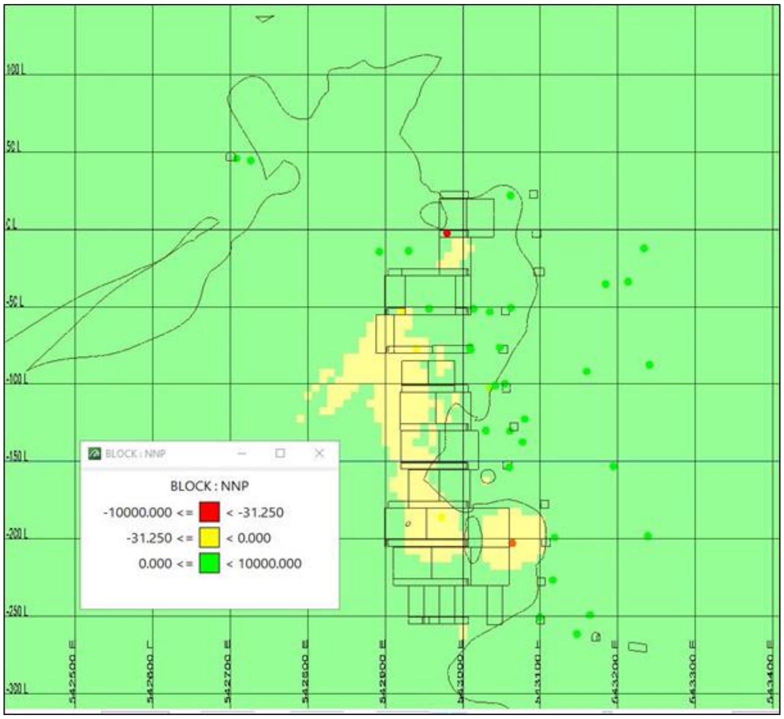
Source: SRK

Figure 14-28: Estimated AP (Cross Section View to Northeast, Stope and Lithology Outlines are Shown)



Source: SRK

Figure 14-29: Estimated NP (Viewing Northeast, Stope and Lithology Outlines are shown)



Source: SRK

Figure 14-30: Estimated NNP (Viewing Northeast, Stope and Lithology Outlines are shown)

Table 14-18: AP & NP Database Characteristics

Data Set	Lithology	Data Range Utilized in Model	Number of Samples Included	Average Value of Utilized Data
AP	Metasediment	0.3 - 75.9	37	13.9
	Metavolcanic	0.3 - 18.8	41	2.0
NP	Metasediment	0.1 - 74.6	37	29.6
	Metavolcanic	4.2 - 63.3	41	32.7

Source: SRK, 2017

Table 14-19: General Estimation Parameters

Lithology	Estimation Pass	Min/Max No. of Samples	Search Orientation (bearing, plunge, dip)	Search Distance [X, Y, Z](m)
Metasediment and Metavolcanic Upper Zone	1	2/5	345, -45, 0	35, 35, 10
	2			75, 75, 25
	3			100, 100, 35
Metasediment and Metavolcanic Upper Zone	1	2/5	345, -90, 0	35, 35, 10
	2			75, 75, 25
	3			100, 100, 35

Source: SRK, 2017

Table 14-20: Estimation Parameter Results

Lithology	Estimation Pass	Number of Samples Used	Average Distance (m)	Percentage of Total Blocks Estimated
Metasediment	1	2.2	20	6
	2	3.3	43	64
	3	2.5	65	30
	All	**2.9**	**48**	**100**
Metavolcanic	1	2.1	20	5
	2	3.0	46	47
	3	2.6	68	48
	All	**2.8**	**55**	**100**

Source: SRK, 2017

Table 14-21: Statistical Validation Results

Lithology	Sample AP	Block AP	% Difference Block to Sample
Metasediment	13.9	13.9	0.1
Metavolcanic	2.0	2.0	0
Lithology	**Sample NP**	**Block AP**	**% Difference Block to Sample**
Metasediment	29.6	26.8	9.3
Metavolcanic	32.7	31.8	2.9

Source: SRK, 2017

14.3 Open Pit and Underground Combined Mineral Resource Statement

Table 14-22 presents the combined open pit and underground resource statement for Haile.

Table 14-22: Combined Open Pit and Underground Resource statement

Mining Type	Resource Category	Au Cut-Off (g/t)	Tonnes (Mt)	Au (g/t)	Au Contained (Moz)
Open Pit	Measured	0.45	7.06	1.97	0.45
Underground	Measured	-	-	-	-
Subtotal	*Measured*	-	*7.06*	*1.97*	*0.45*
Open Pit	Indicated	0.45	52.2	1.63	2.73
Underground	Indicated	1.17	2.71	5.68	0.49
Subtotal	*Indicated*	-	*54.9*	*1.83*	*3.22*
Subtotal	**M & I**		**61.9**	**1.84**	**3.67**
Open Pit	Inferred	0.45	11.1	1.4	0.49
Underground	Inferred	1.17	1.2	5.0	0.20
Subtotal	**Inferred**		**12.3**	**1.7**	**0.69**

Source: SRK, 2016

- The open pit resource is as of January 1, 2017. The underground resource is as of November 1, 2016.
- The open pit Mineral Resources were estimated by Bruce van Brunt, a Qualified Person. The underground Mineral Resources were estimated by Bart Stryhas, a Qualified Person.
- Mineral Resources are not ore reserves and do not have demonstrated economic viability.
- All figures rounded to reflect the relative accuracy of the estimates.
- Underground CoG is based on a mining cost of US$35/t, milling cost of US$10/t, administration cost of US$5.60/t, a gold price of US$1,500/oz. and gold recovery of 90%.
- Open pit CoG is 0.45 g/t Au and US$1,500/oz, reported within a US$1,500 optimized shell.
- Mineral Resource includes the Mineral Reserve.

15 Mineral Reserve Estimate

Separate Mineral Reserve estimates were generated for the open pit and underground mines. A combined Mineral Reserve statement is provided in Section 15.3. The open pit and underground mining areas are located entirely on land owned by HGM. There are no royalties.

15.1 Open Pit Mineral Reserve Estimate

Open pit LoM plans and resulting open pit Mineral Reserves are determined based on a gold price of US$1,300/oz Au. Reserves stated in this report are dated effective as of January 1, 2017.

The ore material is converted from Mineral Resource to Mineral Reserve based primarily on positive cash flow pit optimization results, pit design and geological classification of Measured and Indicated resources. The in situ value is derived from the estimated grade and certain modifying factors.

The open pit reserve consists of several pit areas. Material is truck hauled from the pits to an existing crusher/processing facility. Waste material is categorized and truck hauled to the appropriate waste rock stockpile location.

15.1.1 Introduction

The sub-blocked geological model was regularized to a 5 m x 5 m x 5 m block model. This block size is currently considered appropriate for the loading units operating at Haile. The dilution that was introduced by this process resulted in a 1% increase in ore tonnes and a 2% reduction in gold grade. No further ore losses or ore dilution were applied.

The open pit Ore Reserves are reported within a pit design based on open pit optimization results. The optimization included Measured, Indicated and Inferred Mineral Resource categories with a gold price of US$1,300/oz Au. The reserve pit used to define reserves was based on a US$1,150/oz Au pit shell as the basis of mine design. Subsequent to pit optimization, inferred material (approximately 13%) within the reserve pit was treated as waste and given a zero gold grade. Whittle optimization parameters are shown in Table 15-1.

The overall pit slopes used for the design are based on operational level geotechnical studies and range from 45°to 70°.

Table 15-1: Whittle Optimization Parameters

Parameter	Unit	Value
Base Mining Cost	US$/t	1.37
Incremental Mining Cost	US$/t / 5 m bench	0.00875
PAG Rehabilitation Cost	US$/t PAG waste	0.60
Processing Cost	US$/t ore	10.61
G&A Cost	US$/t ore	3.184
Gold Recovery	%	(1-(0.2152*au grade^-0.3696))
Mill Throughput	Mtpa	4.0
Gold Price	US$/oz	1,300
Gold Refining & Selling Cost	US$/oz	3.00
Royalties	%	0.0
Discount Rate	%	5.0

Source: OceanaGold

A 3D mine design, based on the selected Whittle pit, was completed using Vulcan software and is the basis for the open pit reserves.

15.1.2 Conversion Assumptions, Parameters and Methods

The conversion of Mineral Resource to Mineral Reserve entails the evaluation of modifying factors that should be considered in stating a Mineral Reserve. Table 15-2 shows a reserve checklist and associated commentary on the risk factors involved for the Haile Open Pit Reserve statement.

Table 15-2: Haile Open Pit Reserve Checklist

Unit	Data Evaluated	Data Not Evaluated	Not Applicable	Notes
Mining				
Mining Width	X			Average 300 to 400 m
Open Pit and/or Underground	X			Open pit and underground
Density and Bulk handling	X			Density and swell considered
Dilution	X			SMU 5 m x 5 m x 5 m
Mine Recovery	X			Full mine recovery assumed
Waste Rock	X			NAG/PAD waste dumps
Grade Control	X			Operating mine – blast chips
Processing	X			
Representative Sample	X			Previous feasibility study and operating mine
Product Recoveries	X			Feasibility study - operating
Hardness (Grindability)	X			Feasibility study - operating
Bulk Density	X			Feasibility study - operating
Deleterious Elements	X			Feasibility study - operating
Process Selection	X			Feasibility study - operating
Geotechnical/Hydrological	X			
Slope Stability (Open Pit)	X			Majority FoS greater than 1.3. Small foliation/orientation risk in localized areas.
Water Balance	X			Full site water balance
Area Hydrology	X			Hydrology considered
7Seismic Risk	X			Low
Environmental				
Baseline Studies	X			Operating Mine
Tailing Management	X			Operating Mine
Waste Rock Management	X			Operating Mine – NAG/PAG plan
ARD Issues	X			Lined waste facilities
Closure and Reclamation Plan		X		
Permitting Schedule	X			Ongoing – reasonable expectation of success
Location and Infrastructure				
Climate	X			High rainfall events
Supply Logistics	X			Operating Mine
Power Source(S)	X			Operating Mine
Existing Infrastructure	X			Operating Mine
Labor Supply and Skill Level	X			Operating Mine – Training ongoing
Marketing Elements or Factors				
Product Specification and Demand	X			Gold Market
Off-site Treatment Terms and Costs	X			Favorable refining conditions
Transportation Costs	X			Low
Legal Elements or Factors		X		
Security of Tenure	X			Operating Mine
Ownership Rights and Interests	X			Operating Mine

Unit	Data Evaluated	Data Not Evaluated	Not Applicable	Notes
Environmental Liability	X			ARD potential
Political Risk (e.g., land claims, sovereign risk)	X			Low political risk - USA
Negotiated Fiscal Regime		X		
General Costs and Revenue Elements or Factors				
General and Administrative Costs	X			Operating Mine
Commodity Price Forecasts	X			US$1,300/oz Au
Foreign Exchange Forecasts			X	
Inflation	X			Small
Royalty Commitments	X			No royalty
Taxes	X			Operating Mine
Corporative Investment Criteria		X		
Social Issues				
Sustainable Development Strategy	X			Environmental Impact Statement (EIS)
Impact Assessment and Mitigation	X			EIS
Negotiated Cost/Benefit Agreement	X			EIS
Cultural and Social Influences	X			EIS

Source: SRK, 2017

15.1.3 Reserve Estimate

Mineral reserves were classified using the 2014 CIM Definition standards. Measured Mineral Resources were converted to Proven Mineral Reserves, and Indicated Mineral Resources were converted to Probable Mineral Reserves by applying the appropriate modifying factors, as described herein, to potential mining pit shapes created during the mine design process.

The open pit mine design process results in open pit mining reserves of 55.0 Mt with an average grade of 1.71g/t. The Mineral Reserve statement, as of January 1, 2017, for the Haile Open Pit is presented in Table 15-3.

Table 15-3: **Haile Open Pit Mineral Reserves Estimate as of January 1, 2017**

Description	Tonnes (Mt)	Au (g/t)	Au Contained (Moz)
Proven	7.55	1.97	0.48
Probable	47.5	1.66	2.54
P+P	55.0	1.71	3.02

Source: SRK

- Reserves are based on a US$1,300/oz Au gold price.
- Open pit reserves are stated using a 0.45 g/t cut-off and assume full mine recovery.
- Open pit reserves are not diluted (Further to dilution inherent to the resource model and assume selective mining unit of 5 m x 5 m x 5 m).
- Metallurgical recoveries are based on a recovery curve (1-(0.2152*au grade^-0.3696)) that equate to an overall recovery of 82%.
- Reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- The open pit reserves are valid as of January 1, 2017.
- Mineral Reserves were estimated by Bret C. Swanson, BE (Min) MMSAQP #04418QP, a Qualified Person.

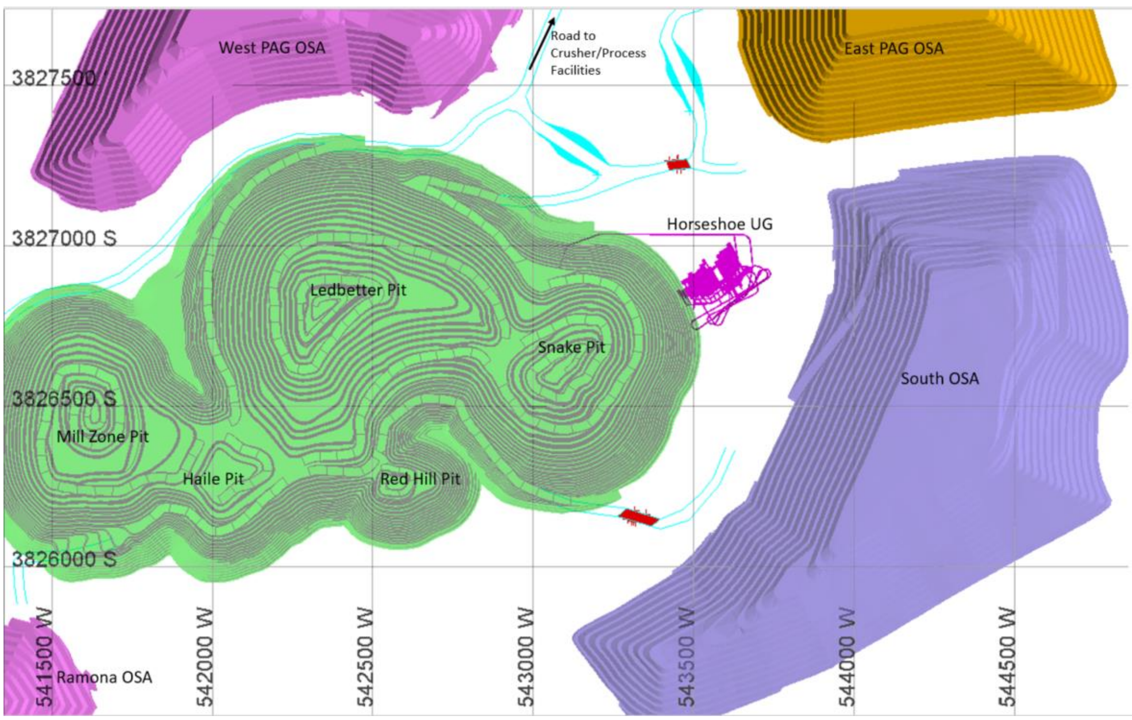
Relevant Factors

SRK knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors that might materially affect the Mineral Reserve estimate.

15.2 Underground Mineral Reserve Estimate

15.2.1 Introduction

Mineral Resources extend below and outside of the existing open pit mine. A portion of these Mineral Resources will not be mined by the ultimate pit shell that is described in this feasibility study and therefore have been evaluated for underground mining. The Mineral Resource area evaluated for underground mining is referred to as “Horseshoe”. Horseshoe is located to the northeast of the Snake Pit, as shown in Figure 15-1.



Source: SRK

Figure 15-1: General Site Layout and Location of the UG Reserve Area

15.2.2 Conversion Assumptions, Parameters and Methods

Measured and Indicated Mineral Resources were converted to Proven and Probable Mineral Reserves by applying the appropriate modifying factors, as described herein, to potential mining block shapes created during the mine design process.

Based on the orientation, depth, and geotechnical characteristics of the mineralization, a transverse sublevel open stoping method (longhole) with ramp access is used. The stopes will be 15 m wide and stope length will vary based on mineralization grade and geotechnical considerations. A spacing of 25 m between levels is used. Cemented rock fill (CRF) will be used to backfill 75% of the stopes and non-cemented waste rock will be used in the remaining stopes. The CRF will have sufficient strength to allow for mining adjacent to backfilled stopes.

For the FS design, the deposit has been divided into two major production areas. The “upper area”, which extends from the -70L to the top of the orebody, will be mined from bottom to top, with stoping initially commencing on the -70L. A permanent sill pillar has been designed between the -70L and -75L. The “lower area” will also be mined from bottom to top, with stoping commencing on the -240L and progressing upward until the permanent sill pillar on the -75L is reached.

All Mineral Reserve tonnages are expressed as "dry” tonnes (i.e., no moisture) and are based on the density values stored in the block model. Inferred Mineral Resources are not included in the mine plan. Mining dilution and recovery have been applied to the reserves using the methodologies described in the following sections.

Dilution

The mining dilution estimate for the FS is based on the equivalent linear overbreak/slough methodology (Clark, 1997). ELOS is an empirical design method that is used to estimate the amount of overbreak/slough that will occur in an underground opening based on rock quality and the hydraulic radius of the opening. For the FS design, ELOS was applied to in situ rock exposures and to the CRF backfill walls wherever mining will occur adjacent to a secondary stope. In addition to ELOS allowances, a 0.5% additional dilution allowance was used to account for other potential sources of dilution (e.g., dilution from the floor when mucking a stope).

ELOS assumptions are shown in Table 15-4.

Table 15-4: Dilution ELOS Assumptions

Type	ELOS Value (m)
Sidewalls (rock)	0.50
Sidewalls (backfill)	0.25
Endwalls (rock & backfill)	0.15
Bottom (backfill)	0.05

Source: SRK

The rock sidewall/endwall dilution material will contain low-grade mineralization. To determine the grade that should be used for dilution material, 3D triangulations were generated showing the dilution for a vertical section through the entire deposit. The grade for the diluting material was calculated from the block model and averaged 1.45 g/t Au.

Based on the FS design, stopes are located above other stopes 66% of the time, and 25% of the time stopes are not adjacent to other stopes. Based on these percentages, in areas where dilution will carry grade the 1.45 g/t Au value was applied. A zero grade is used for backfill dilution.

The ELOS and additional dilution factor of 0.5% gives a total dilution of 7%, with 3% of the dilution having grade (of 1.45 g/t Au) from in situ rock that is exposed in the stopes.

For horizontal development in ore (other than drill drives inside a stope), dilution of 5% was applied at zero grade.

Recovery

A stope recovery factor of 93% was used. The following items were used to calculate this factor:

- Material loss into backfill (floor) of 0.15 m;
- Material loss to side and endwalls (under blast) of 0.15 m;
- Material loss to stabilizing pillars (limited application, only as needed);
- Material loss to mucking along the sides and in blind corners of the stopes; and
- Additional loss factor due to rockfalls, misdirected loads, and other geotechnical reasons.

A development recovery factor of 95% was used for all horizontal development.

Additional Allowances Factors

Additional ramp allowance factors were used to account for additional excavations not included in the FS design. These items will be designed at the detailed planning stage. Items are summarized in Table 15-5 for the main ramps and in Table 15-6 and Table 15-7 for the footwall accesses.

Table 15-5: Main Ramp Additional Allowance Factors

Main Ramps – 5 x 5.5 (arched back)	Allowance Factors (m^3)
Muck Bays - 4.5 m x 4.5 m x 16 m long, 2 per 500 m of ramp	648
Muck Bays - high backs for loading - 7 m tall	268
Drill Bays – re-purpose muck bays for drilling	-
Electrical - 4.5 m x 4.5 m x 4 m long, 1 per 500 m ramp	81
Pump station - 4.5 m x 4.5 m x 16 m long, 1 per 500 m of ramp	324
Passing bays - 4 m wide, 35 m long, 4.5 m high, 1 per 500 m of ramp	685
Total additional development volume	2,006
500 m ramp volume	12,670
Additional Allowance for main ramps:	16%

Source: SRK

Table 15-6: Footwall Access Additional Allowance Factors – Upper Area

Footwall Access on levels	Allowance Factors (m^3)
Pump station – none – drain back to main ramp	-
Electrical - 4.5 m x 4.5 m x 4 m long, 1 per level	81
Muck Bays (1) - high backs for loading - 7 m tall	122
Total additional development volume	203
Footwall Access level volume (2)	3,750
Additional allowance for main ramps:	5%

Source: SRK
(1) One muckbay included in the design per level
(2) Upper mining block levels are 150 m long

Table 15-7: Footwall Access Additional Allowance Factors – Lower Area

Footwall Access on levels	Allowance Factors (m^3)
Pump station – none – drain back to main ramp	-
Electrical - 4.5 m x 4.5 m x 4 m long, 1 per level	81
Muck Bays (1) - high backs for loading - 7 m tall	122
Total additional development volume	203
Footwall Access level volume (2)	1,750
Additional Allowance for main ramps:	12%

Source: SRK
(1) One muck bay included in the design per level
(2) Upper mining block levels are 70 m long

These factors are applied to the main ramp and footwall accesses length and the volume/tonnage of rock is increased similarly. This captures the development time required to excavate these openings, cost to do so, and reports the expected amount of waste volumes for tracking purposes.

15.2.3 Reserve Estimate

Mineral Reserves were classified using the 2014 CIM Definition standards. Indicated Mineral Resources were converted to Probable Mineral Reserves by applying the appropriate modifying factors, as described herein, to potential mining shapes created during the mine design process.

The underground mine design process resulted in underground mining reserves of 3.1 Mt (diluted) with an average grade of 4.38 g/t Au. The Mineral Reserve statement, as of January 1, 2017, for the Haile Horseshoe Underground is presented in Table 15-8.

Table 15-8: **Haile Horseshoe Underground Reserves Estimate as of November 1, 2016**

Description	Tonnes (mt)	Au (g/t)	Au Contained (Moz)
Proven	-	-	-
Probable	3.12	4.38	0.44
P+P	3.12	4.38	0.44

Source: SRK

- Reserves are based on a gold price of US$ 1,300/oz. Metallurgical recoveries are based on a recovery (1-(0.2152*au grade^-0.3696)) that equates to an overall recovery of 88%.
- Underground reserves are stated using a 1.5 g/t cut-off. Mining recovery ranges from 93% to 100% depending on activity type. Mining dilution is applied, partially at zero grade and partially using a grade calculated from a representative section. The dilution ranges from 0% to 7% depending on activity type.
- Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- Mineral Reserves have been stated on the basis of a mine design, mine plan, and cash-flow model.
- The underground Mineral Reserves are effective as of November 1, 2016.
- The Mineral Reserves were estimated by Joanna Poeck, BEng Mining, SME-RM, MMSAQP #01387QP, a Qualified Person.

Relevant Factors

SRK knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors which could materially affect the underground Mineral Reserve estimate.

15.3 Open Pit and Underground Combined Reserves Statement

Table 15-9 presents the combined open pit and underground Mineral Reserves statement for Haile.

Table 15-9: Reserve Statement for OceanaGold's Haile Gold Mine

Mining Type	Resource Category	Tonnes (Mt)	Au (g/t)	Au Contained (Moz)
Open Pit	Proven	7.55	1.97	0.48
Underground	Proven	-	-	-
Subtotal	Proven	7.55	1.97	0.48
Open Pit	Probable	47.50	1.66	2.54
Underground	Probable	3.12	4.38	0.44
Subtotal	Probable	50.6	1.83	2.98
Total (OP + UG)	**P+P**	**58.2**	**1.85**	**3.46**

Source: SRK

- The open pit reserve is as of January 1, 2017. The underground reserve is as of November 1, 2016.
- Mineral Reserves are based on a gold price of US$ 1,300/oz. Metallurgical recoveries are based on a recovery curve (1-(0.2152*au grade^-0.3696)) that equates to an overall recovery of 82% for the open pit material and 88% for the underground material.
- Open pit Mineral Reserves are stated using a 0.45 g/t Au cut-off and assume full mine recovery. Open pit reserves are not diluted (Further to dilution inherent to the resource model and assume selective mining unit of 5 m x 5 m x 5 m)
- Open pit reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- Underground Mineral Reserves are stated using a 1.5 g/t Au cut-off. Mining recovery ranges from 93% to 100% depending on activity type. Mining dilution is applied, partially at zero grade and partially using a grade calculated from a representative section. The dilution ranges from 0% to 7% depending on activity type.
- Mineral Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- Mineral Reserves have been stated on the basis of a mine design, mine plan, and cash-flow model.
- The open pit Mineral Reserves were estimated by Bret C Swanson, BE (Min) MMSAQP #04418QP, a Qualified Person. The underground Mineral Reserves were estimated by Joanna Poeck, BEng Mining, SME-RM, MMSAQP #01387QP, a Qualified Person.

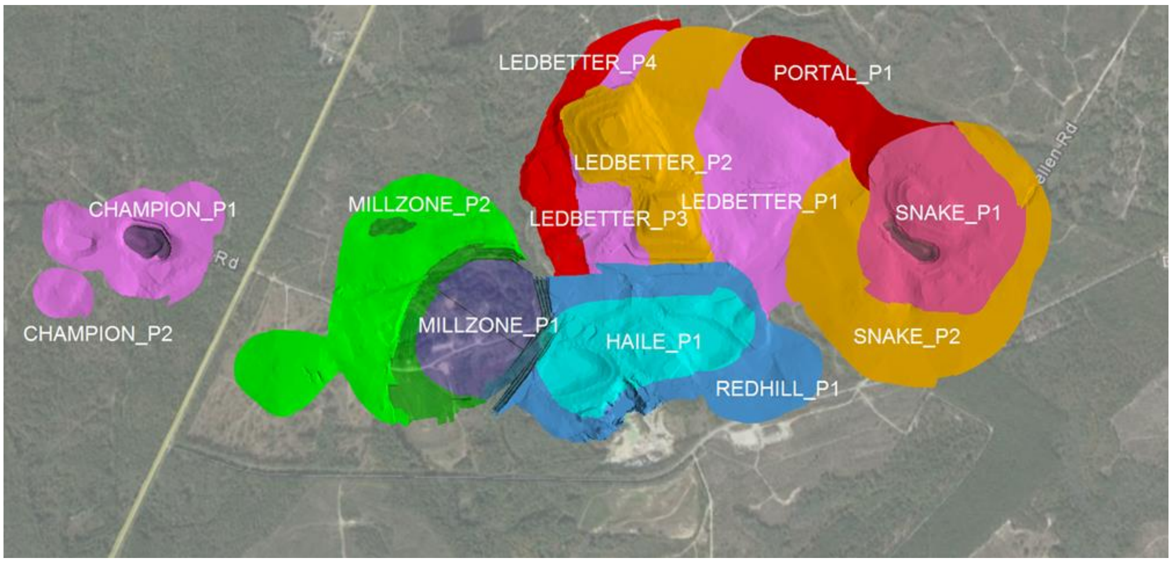
16 Mining Methods

Both open pit and underground mining methods will be used at Haile. As such the following sections describe the open pit and underground mining methods separately. A combined open pit and underground production schedule is provided in Section 16.3.

16.1 Open Pit Mining Methods

SRK worked in conjunction with OceanaGold staff in the mine planning of the Haile open pit operations. OceanaGold was responsible for pit optimization and mine fleet capital and operating cost estimation while SRK was responsible for pit design, phase design and production scheduling. The mine plans are valid from January 1, 2017 and are based on a surveyed topographical surface of the same date. During the first six months of 2017, mining was continuing at the Mill Zone and pre-stripping of the Snake pit was commencing. The optimized mine plan looks to maximize potential reserves from open pit operations while integrating the Horseshoe underground mine into the Haile LOM plan and reserve.

The primary pit names referenced in the sections to follow are based on historical naming conventions when many of the smaller gold pits did not merge into the large Haile open pit described in this report. As such, phases have been named to replicate the historical pit areas. Figure 16-1 illustrates the pit area names



Source: SRK

Figure 16-1: Haile Open Pit Naming Convention

16.1.1 Current or Proposed Mining Methods

Haile is currently being mined using conventional open pit methods. These procedures will not change with the expansion of the open pit that is proposed in this report.

The material encountered at Haile is a combination of soft (Costal Plains Sands [CPS] and saprolite) and hard (metavolcanics and metasediments) rock units.

As the name suggests, CPS is loosely consolidated sand which can be mined without the need for drilling and blasting. Mineralization is not present in CPS thus drilling for the purposes of ore control and waste definition is not necessary.

Saprolite is free dug where possible, however blasting is required near the hard rock contact. Mineralized zones are drilled for ore control sampling. Haile's Overburden Management Plan (OMP) requires waste definition sampling of saprolite within 15 m of the base of weathering.

Drilling and blasting is required in all hard rock. Ore zones are drilled with 115 mm holes to a depth of 5 m (plus 0.5 m sub drill) and waste zones are drilled with 171 mm holes to a depth of 10 m (plus one1 m sub drill). Holes are sampled twice (top half and bottom half) for ore control and waste definition, the sub drill portion of the hole is not sampled.

Ore is mined on 5 m benches in two 2.5 m flitches and waste is mined on 10 m benches in two 5 m flitches. Loading is done with hydraulic excavators/shovels and front-end loaders. The haul truck fleet is a mix of and 90-t and 135-t trucks.

16.1.2 Parameters Relevant to Mine or Pit Designs and Plans

Geotechnical

Geotechnical open pit slope design parameters, shown in Table 16-1, were developed by BGC Engineering Inc. (BGC) for the current study. The basis of the slope design parameters is:

- The geological model for the mine developed by OceanaGold (as described in Section 14.1)
- A geotechnical unit model developed by BGC
- A structural geology model developed by BGC
- Slope stability assessments completed by BGC

The rock mass and structural geology models for the open pit slope designs are based on data collected in 2010, 2013, and 2016 (Table 16-2). The dataset consists of:

- Core logging, oriented core, and/or televiewer data from 18 drill holes in the area of the proposed open pits.
- Laboratory tests of uniaxial compressive strength (19 tests), triaxial compressive strength (7 tests), and direct shear strength (DSS) (12 tests) of rock core samples and natural discontinuities sampled from the drill core.
- Mapping and pit slope observations from existing operational slopes in the Mill Zone Pit.

The geotechnical unit model consists of eight units, based on geology and material properties:

1. Coastal Plain Sands (CPS).
2. Saprolite (SAP) derived from highly weathered bedrock.
3. Weathered dikes (WDD) mainly of diabase.
4. Fresh dikes (FDD) mainly of diabase.
5. Weathered metavolcanic (WMV).
6. Fresh metavolcanic (FMV).
7. Weathered metasediment (WMS).
8. Fresh metasediment (FMS).

The gradational contacts between saprolite to weathered rocks and weathered rocks to fresh rocks are key features of the geotechnical unit model. The approximate contact between the saprolite and weathered rock is provided by the OceanaGold geological model. BGC has estimated the average elevation of the transition from weathered to fresh rock as 70 m ASL, across the open pit area. In general, the shear strengths of the geotechnical units increase with mining depth. The CPS and SAP units are soil-like. The weathered rocks are generally medium strong and fair quality. The fresh dikes and metavolcanic rocks are generally strong and good quality. The fresh metasediments rocks are generally medium strong and fair to good quality.

The structural geology model for the open pit consists of two domains representing the parts of the mine expected to generally comprise metasedimentary or metavolcanic rocks. The main structural geology fabrics that affect the pit slope designs are:

- Regional northwest dipping foliation, best developed in the metasedimentary rocks;
- A southeast dipping joint set; and
- Sub-vertical or steeply dipping discontinuities at or parallel to the diabase dike contacts.

Fault orientations have been inferred from regional mapping data; there is no three-dimensional fault model for the open pit area. The faults, foliation, and joint sets of the structural geology model are the most important control on the achievable bench face and inter-ramp slope angles in the weathered and fresh rocks of Haile.

Kinematic and limit equilibrium slope stability analyses were completed to develop the recommended bench and inter-ramp slope design parameters. The design parameters were developed for individual slope sectors defined by geotechnical units, structural domains, and expected pit slope orientations. The key pit slope design recommendations are:

- Inter-ramp slope angles vary from 37° to 50° in rock (FMS, WMS, FMV, WMV, FDD and WDD), are 30° in SAP and 27° in CPS.
- Full depressurization of the weathered rock geotechnical units (WMS and WMV) is required on the north walls (Sectors WMV-345 and WMS-345) and south walls (Sectors WMV-163 and WMS-163) of the pits. Depressurization of the FMS geotechnical unit is required for the south walls (FMS-163) of the pit. The depressurization of these slopes may need to be accomplished via in-pit wells or horizontal drains. Further work is required to design the dewatering and depressurization system for the planned pit.

The ultimate pit presented here-in suitably conforms to the bench and inter-ramp design recommendations provided by BGC Engineering Inc. (BGC). The overall slopes meet the industry standard factor of safety (FOS) and/or probability of failure (POF) stability criteria targets, provided the recommended levels of slope depressurization are achieved and control blasting is implemented on final walls. The overall slope stability analysis cross-sections (Figure 16-2) include zones of disturbed rock mass to account for the effects of blasting and stress relief, strength anisotropies reflecting the dominant rock mass fabric of the structural geology model, and geotechnical unit properties and boundaries from the geological and geotechnical unit model. Further details of the overall slope analyses are provided in BGC, 2017.

The main controls on the stability of Haile overall pit slopes are geological structures and water pressures. The north and south walls represent areas were more favorable structural geology conditions could result in increased slope angles. If unfavorable geological structures are identified in the east or west walls through future work, these slopes may require flattening. Future geological and geotechnical work should focus on the development of a 3D fault model with a level of detail suitable for pit slope stability assessments. Further work to design and optimize a pit dewatering/depressurization plan is also required.

Table 16-1: Open Pit Slope Design Parameters

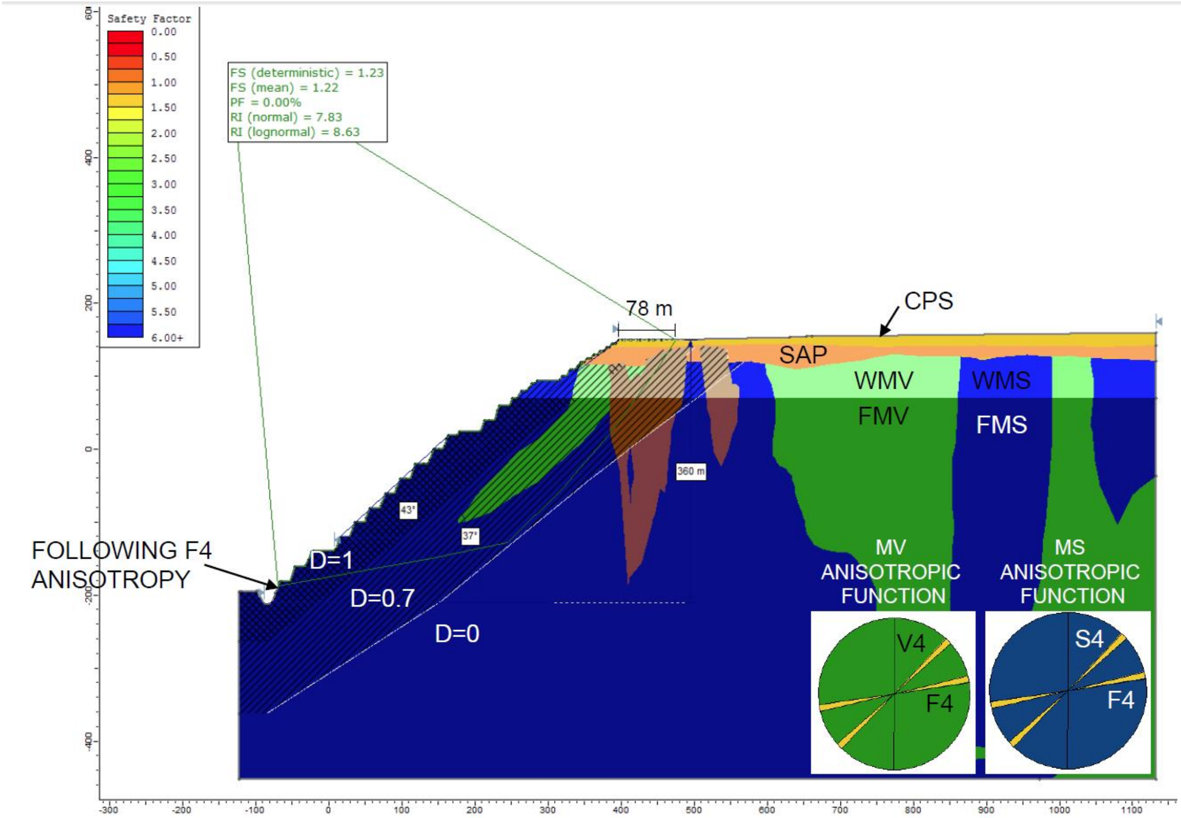
Geotechnical Domain	Sector Azimuth	Bench Height	Face Angle	Berm Width	Inter-Ramp
Coastal Plain Sand	All	5.0 m	45°	5.0 m	27°
Saprolite	All	5.0 m	55°	5.0 m	30°
Weathered Metavolcanics	005° to 035°	10.0 m	65°	6.5 m	42°
	035° to 120°	10.0 m	65°	6.5 m	42°
	120° to 205°	10.0 m	55°	6.5 m	37°
	205° to 325°	10.0 m	65°	6.5 m	42°
	325° to 005°	10.0 m	65°	6.5 m	42°
Weathered Metasediments	005° to 035°	10.0 m	65°	6.5 m	42°
	035° to 120°	10.0 m	65°	6.5 m	42°
	120° to 205°	10.0 m	55°	6.5 m	37°
	205° to 325°	10.0 m	65°	6.5 m	42°
	325° to 005°	10.0 m	65°	7.7 m	39°
Fresh Metavolcanics	005° to 035°	20.0 m	70°	9.5 m	50°
	035° to 120°	20.0 m	70°	9.5 m	50°
	120° to 205°	20.0 m	60°	9.5 m	44°
	205° to 325°	20.0 m	70°	9.5 m	50°
	325° to 005°	20.0 m	70°	9.5 m	50°
Fresh Metasediments	005° to 035°	20.0 m	70°	13.4 m	44°
	035° to 120°	20.0 m	70°	9.5 m	50°
	120° to 205°	20.0 m	60°	9.5 m	44°
	205° to 325°	20.0 m	70°	9.5 m	50°
	325° to 005°	20.0 m	70°	17.4 m	39°

Source: BGC Engineering Inc., 2017

Table 16-2: Open Pit Geotechnical Drill Holes

Hole ID	Geotechnical Log By:	Year	Drilling Method	Easting (1)	Northing (1)	Elevation (1)	Trend (°)	Plunge (°)	Length (m)	Available Data (2)
DDH612	SRK	2016	Triple Tube Coring	542943	3826918	138	289	-71	152	TV, OC, GL, RM
DDH613	SRK	2016		542441	3827094	156	057	-70	152	TV, OC, GL, RM
DDH614	SRK	2016		541882	3826620	133	241	-70	151	TV, OC, GL, RM
DDH342	OceanaGold Staff	2010		543226	3826774	158	354	-73	305	OC, GL, RM
DDH343	OceanaGold Staff	2010		543129	3826654	149	181	-64	305	OC, GL, RM
DDH344	OceanaGold Staff	2010		542645	3826871	148	000	-65	305	OC, GL, RM
DDH345	OceanaGold Staff	2010		542630	3826834	146	183	-65	306	OC, GL, RM
DDH346	OceanaGold Staff	2010		542222	3826330	146	130	-62	122	OC, GL, RM
DDG347	OceanaGold Staff	2010		541748	3826360	146	180	-62	183	OC, GL, RM
DDH350	OceanaGold Staff	2010		541712	3826380	148	000	-58	184	OC, GL, RM
DDH351	OceanaGold Staff	2010		542249	3826550	138	001	-54	268	OC, GL, RM
PW-1301	-	2013	Reverse Circulation	542970	3826820	141	-	-90	317	TV, GL
PZ-1303D	-	2013		542768	3828140	161	-	-90	261	TV, GL
PZ-1308	-	2013		541251	3826329	161	-	-90	157	TV, GL
PZ-1310	-	2013		541293	3826434	158	-	-90	155	TV, GL
RC-2127	-	2013		542970	3826820	141	-	-90	317	TV, GL
RC-2135	-	2013		541675	3826860	125	-	-90	256	TV, GL
RC-2140	-	2013		541220	3826395	162	-	-90	152	TV, GL

Source: BGC Engineering Inc., 2017
(1) All coordinates provided in UTM NAD 83. All coordinates provided in UTM NAD 83.
(2) TV = televiewer data; OC = oriented core data; GL = geology data; RM = rock mass data.



Source: BGC, 2017
Analysis details are provided in BGC, 2017

Figure 16-2: Example overall analysis cross-section: cross-section "C" for the south wall of the ultimate pit

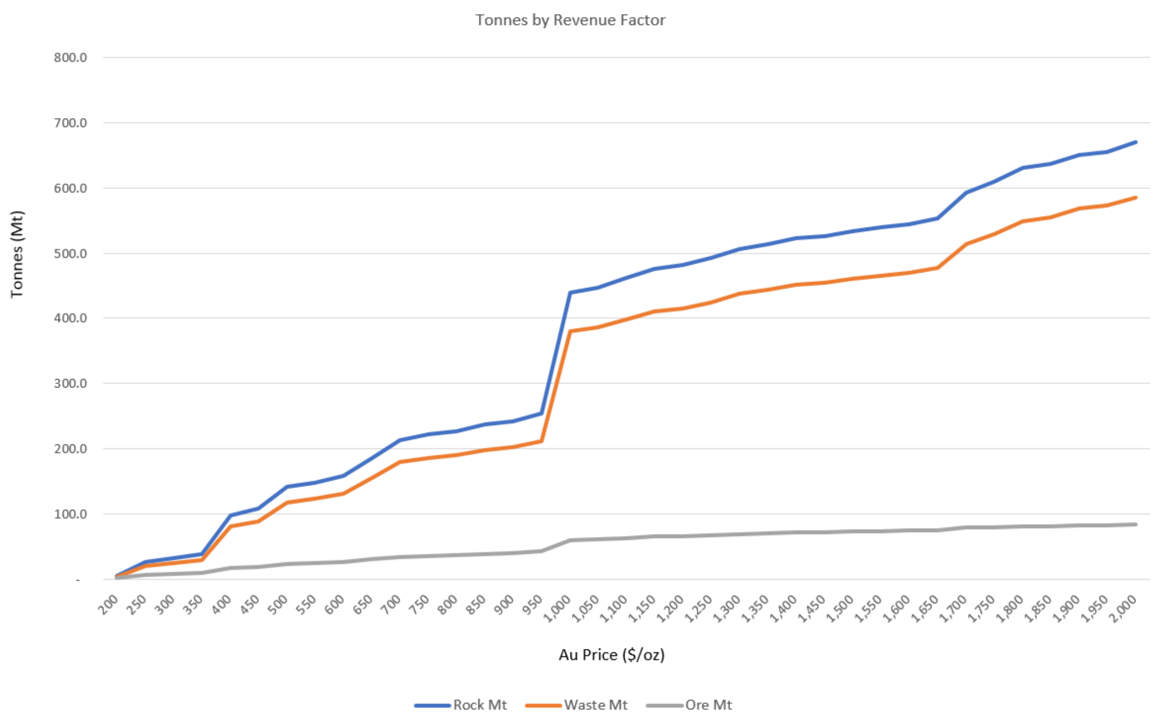
Hydrogeological

Dewatering and depressurization of highwalls is on-going. Ground water levels continue to decrease and the drawdown trend appears to be reaching steady state in active mining areas. There are currently ten dewatering wells in service at Haile. Ten additional wells are planned for the remainder of 2017 and 2018.

16.1.3 Optimization

The sub-blocked geological model was regularized to a 5 m x 5 m x 5 m block model. This block size is considered appropriate for the loading units currently operating at Haile. The dilution that is introduced by this process resulted in a 1% increase in ore tonnes and a 2% reduction in gold grade. No further ore losses or ore dilution were applied.

The open pit Ore Reserves are reported within a pit design based on open pit optimization results (Figure 16-3). The optimization included Measured, Indicated and Inferred Mineral Resource categories with a gold price of US$1,300/oz Au.



Source: OceanaGold

Figure 16-3: Whittle Optimization Tonnes by Revenue Factor

Table 16-3: Optimization Results for Selected Shell

Shell Au price US$/oz	Rock (Mt)	Waste (Mt)	Strip Ratio (W:O)	MI Ore (Mt)	MI Ore (g/t)	MI Ore Contained Oz (koz)
1,150	476	419	7.3	57.1	1.67	3,065

Source: OceanaGold

Whittle optimization parameters are summarized in Table 16-4.

Table 16-4: Whittle Optimization Parameters

Parameter	Unit	Value
Base Mining Cost	US$/t	1.37
Incremental Mining Cost	US$/t / 5 m bench	0.00875
PAG Rehabilitation Cost	US$/t PAG waste	0.60
Processing Cost	US$/t ore	10.61
G&A Cost	US$/t ore	3.184
Gold Recovery	%	(1-(0.2152*au grade^-0.3696))
Mill Throughput	Mtpa	4.0
Gold Price	US$/oz	1,300
Silver Price	US$/oz	19.00
Gold Refining & Selling Cost	US$/oz	3.00
Silver Refining & Selling Cost	US$/oz	0.00
Royalties	%	0.0
Discount Rate	%	5.0

Source: OceanaGold

The base mining cost was applied to all blocks and an incremental cost was added to blocks below the elevation where the haulage ramp exits the pit. The incremental cost was not added to blocks above the pit exit. As the Whittle optimization was completed prior to the pit design, pit exit elevations were taken from previous pit design work.

Rehabilitation costs have been added to potentially acid generating (PAG) blocks that will not be processed as this material will require permanent storage within the lined JPAG facility. The cost associated with permanent storage of processed blocks within the Tailing Storage Facility (TSF) is included in the processing cost. The determination of whether a block is processed and, by extension rehabilitation cost, is made by Whittle during optimization.

16.1.4 Mine Design

Mine block model

SRK was supplied with the OceanaGold Resource block model. For mine planning purposes, the model was modified to include:

- Geotechnical variables for berm width, batter angle and bench height
- Ore and waste classifications based on different gold prices and Measured, Indicated and Inferred material.
- Haul routes for red/yellow waste, green waste, ore and saprolite
- Cycle times and distances for red/yellow waste, green waste, ore and saprolite
- NAG/PAG determination
- Production period flagging

The NAG/PAG determinations govern the routing of waste material to either a lined PAG waste dump facility or an unlined NAG dump.

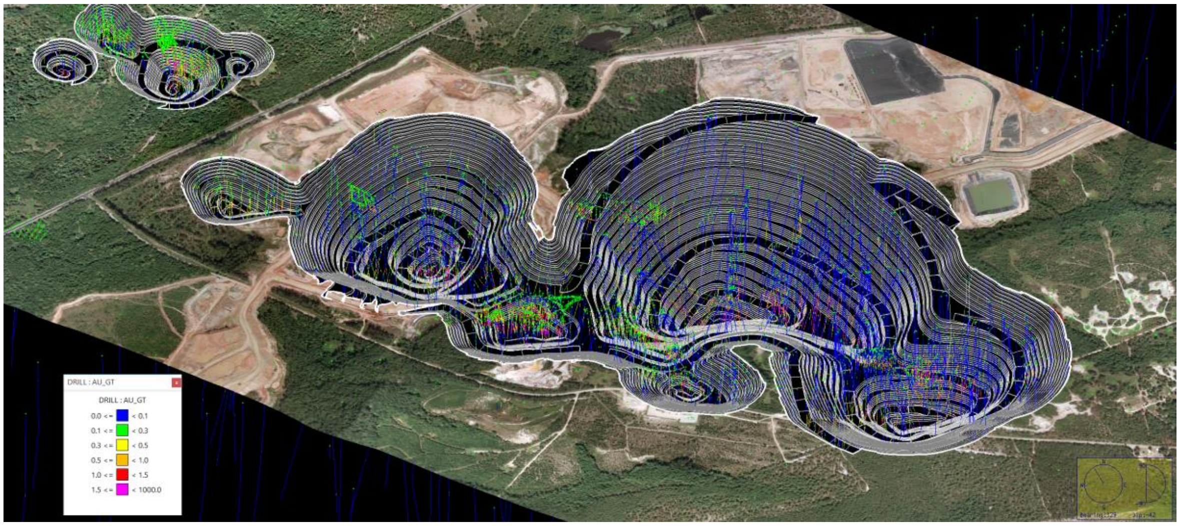
Pit design

SRK was provided with a Whittle pit shell that formed the basis of detailed pit and phase design that formed the basis of reserves. Combined with interim higher value Whittle pit shells, SRK used the Whittle pits as a guide for practical phase design.

The major design parameters used are as follows:

- Ramp grade = 10%,
- Full ramp width = 29m (3x operating width for 789D),
- Single ramp width = 18.5m for up to 60m vertical or 3 benches
- Minimum mining width = 75 m but targets 200 m to 300 m
- Flat switchbacks
- Bench heights, berm widths and bench face angles in accordance with Table 16.1

Figure 16-4 illustrates the final open pit design and associated ramp system. Ramp locations targeted saddle points between the various pit bottoms with ramps also acting as catch benches for geotechnical purposes. Each bench has at least one ramp for scheduling purposes.



Source: SRK 2017

Figure 16-4: LoM Pit design

Grade Tonnage

The grade tonnage data has been limited to Measured and Indicated resources within the SRK reserve pit. The grade tonnage curve shows that as the grade increases from 1.18 g/t through 3.31 g/t Au, the stripping ratio increases from approximately 5:1 to 26:1. While the current reserve pit balances reserve size with mine life, there is the potential to modify the operating parameters to increase and decrease grade according to operational parameters encountered during operations (i.e., a grade binning approach to increase the gold grade is possible by increasing the mining rate and stockpiling lower grade material.)

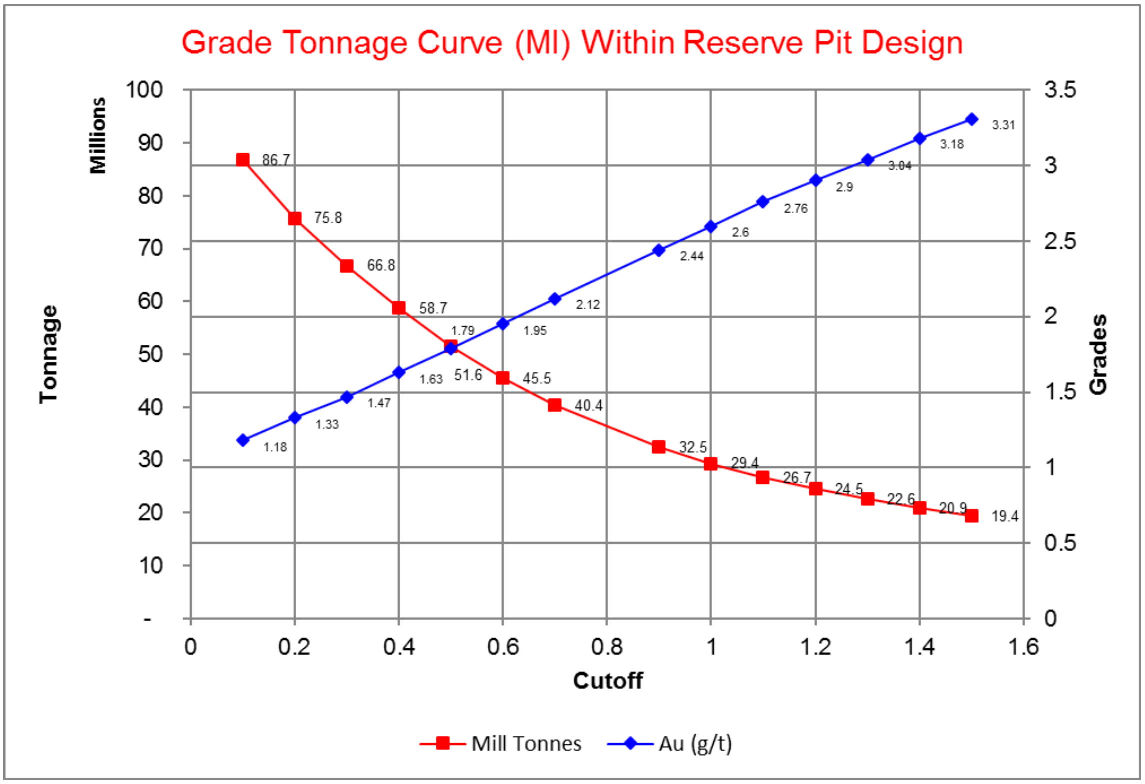
Table 16-5 details the grade tonnage and incremental tonnage at various cut-offs within the pit design. The CoG used for reserves is 0.45 g/t.

Table 16-5: Grade Tonnage Curve within the Pit Design

Au Cut-off (g/t)	Au (g/t)	Mill Tonnes (Mt)	Increment	Inc Grade (g/t)	Inc Tonnage (Mt)	Strip Ratio (W:O)	Contained Gold (Moz)
0	0.19	536.29	0.0	0.00	449.58		
0.1	1.18	86.71	0.1	0.15	10.96	5.18	3.29
0.2	1.33	75.75	0.2	0.25	8.96	6.08	3.24
0.3	1.47	66.79	0.3	0.35	8.07	7.03	3.16
0.4	1.63	58.72	0.4	0.45	7.10	8.13	3.08
0.5	1.79	51.62	0.5	0.55	6.12	9.39	2.97
0.6	1.95	45.50	0.6	0.65	5.14	10.79	2.85
0.7	2.12	40.36	0.7	0.79	7.88	12.29	2.75
0.9	2.44	32.48	0.9	0.95	3.11	15.51	2.55
1.0	2.60	29.37	1.0	1.05	2.69	17.26	2.46
1.1	2.76	26.69	1.1	1.15	2.19	19.10	2.37
1.2	2.90	24.50	1.2	1.25	1.92	20.89	2.28
1.3	3.04	22.58	1.3	1.35	1.72	22.75	2.21
1.4	3.18	20.85	1.4	1.45	1.46	24.72	2.13
1.5	3.31	19.39	1.5	3.31	19.39	26.65	2.06

Source: SRK

Figure 16-5 shows the grade tonnage curve graphically above a 0.1 g/t lower limit.



Source: SRK

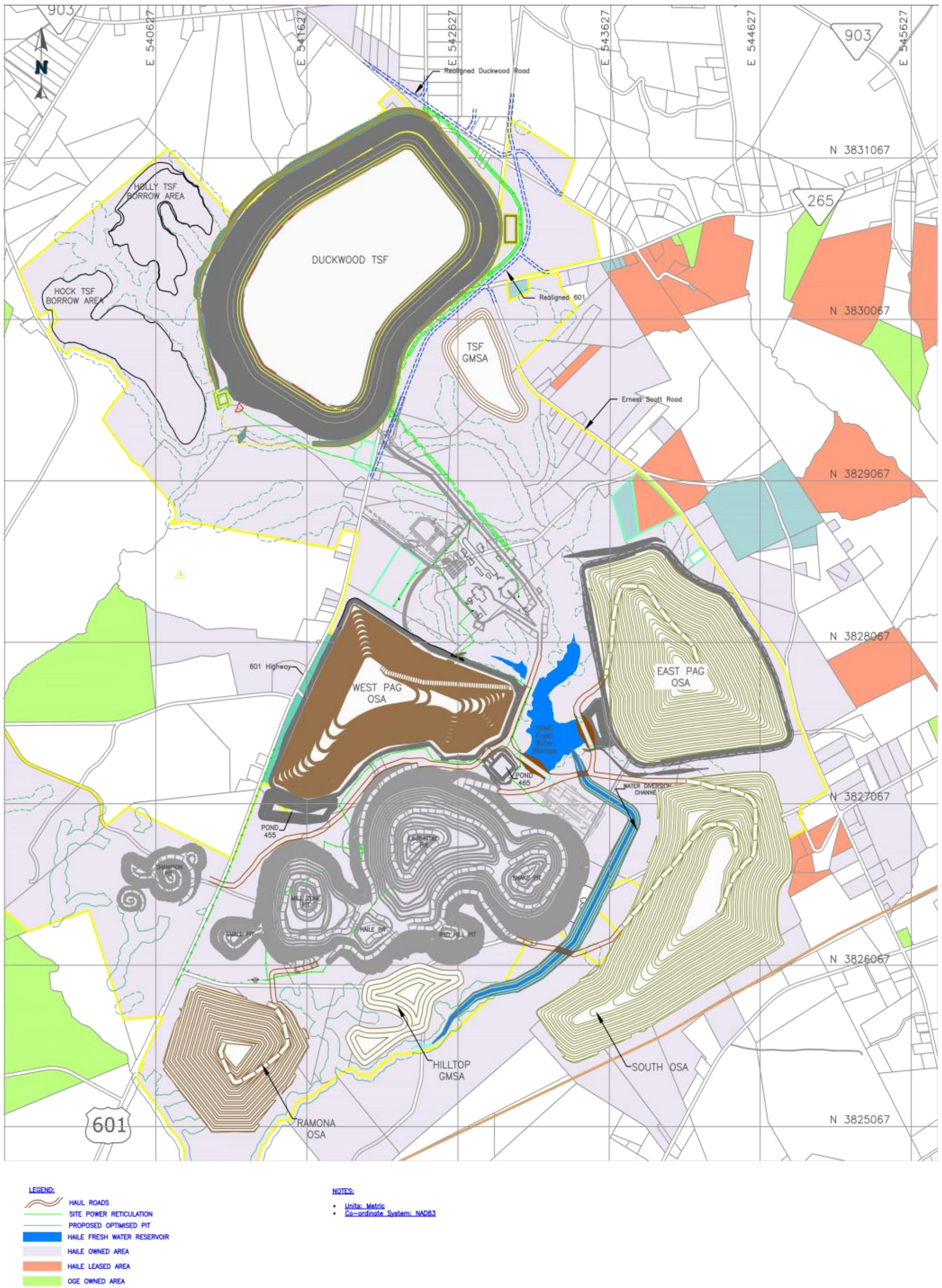
Figure 16-5: Grade Tonnage Curve within Reserve Pit (Measured and Indicated Material)

16.1.5 Overburden

The acid generation potential of the overburden material at Haile has been evaluated. There are three categories of overburden material at Haile: green, yellow and red in order of increasing potential for acid generation. The AP potential informs the placement of the material as follows:

- All red material is sent to the West (early in the mine life) or East (later in the mine life) overburden storage areas. These are lined facilities, with the West already being partially constructed and in use. Material will be placed in lifts, compacted and covered in saprolite.
- Both West and East overburden storage areas will have the lift faces covered in saprolite capping material, as well as the top for closure.
- Yellow material can be stored either in the East or West overburden storage areas, or below a prescribed water table in the pit void. Yellow material in-pit will be mixed with lime before placement in the pit void. Material will be placed in lifts, compacted and covered in saprolite.
- Green material will be stored in unlined facilities or backfilled into the pit void. The angle of repose face of each lift will eventually be dozed to reclamation angle.
- The historic heap leach material, the historic 188 overburden storage, and the historic tailings that will be mined in the pit will be placed on the West overburden storage area.

The overburden storage is discussed in more detail in Section 18.2, with the final year site plan shown in Figure 16-6.



Source: Oceana

Figure 16-6: Final Pit Design and Ultimate Overburden Storage Site Plan

16.1.6 Mine Production Schedule

Cut–off grade

OceanaGold have used a mine breakeven CoG for the determination of ore and waste. The base assumptions for the calculation are detailed in Table 16-6 below. The actual CoG grade applied to Measured and Indicated Resources was rounded to 0.45 g/t Au.

Table 16-6: Cut-off Grade Calculation

		CoG
Assumptions		
Gold Price	US$/oz	$1,300
Gold Price	US$/g	$41.80
Smelting & Refining	US$/oz	$3.00
Au Grade	g/t	0.44
Au Recovery	%	82.5%
Operating Costs		
Smelting & Refining	US$/t ore	$0.0354
Mining	US$/t mined	$1.5
Processing	US$/t ore	$13.79
subtotal	US$/t	$15.3
CoG - Head Grade	g/t	0.445

Source: SRK, 2017

Dilution

Reserves are based on the Haile resource block model that was regularized using a 5 m x 5 m x 5 m block dimension. This method takes sub blocked blocks less than the 5 m dimension and redistributes the gold content to the selective mining unit (SMU) for the mine fleet. The SMU and also approximates the grade control granulation for short term planning.

Phase Design Inventory

The ultimate pit design has been broken into 13 mine phases for sequenced extraction in the SRK production schedule. The design parameters for each phase are the same as those used for the final pit design including assumed ramp widths. Phase designs were constructed by splitting up the final pit into smaller and more manageable pieces, while still ensuring each bench within each phase has ramp access. The phases have been developed by balancing mining constraints with the optimum extraction sequence suggested by pit optimization results presented previously.

The major design considerations include:

- Mill Zone P1 and Snake P1 are low strip ratio phases for early ore exposure that can be mined independently, keep current drainages exposed and allow reasonable mining faces
- Portal P1 is a waste only dropcut that intersects the Snake P1 pit and allows portal access for the Horseshoe underground mine
- Snake P2 is an expansion of Snake P1 where good gold grade is closer to surface but gold does decrease with depth
- Redhill P1 and Haile P1 are moderately profitable phases and provide ore while the Ledbetter phases are mined

- Ledbetter phases P1 to P4 are based on a mining width of approximately 300 m. This allows flexibility in the mine schedule to accelerate mining and maintain a comfortable bench sinking rate.
- Mill Zone P2 and Champion pits are low value phases and are mined at the end of the mine life.

The phases and direction of extraction allow for multiple benches on multiple elevations with a sump always available for pit dewatering. This means that during periods of heavy rainfall, perched benches will be available for extraction.

Once the phases have been designed, solid triangulations are created for each phase as they cut into topography from previous phases. These solid phases are then shelled (cut) on a 5 m lift height that corresponds to one block model block. These shells form a bench within each phase and represent the basic unit that is scheduled for the life-of-mine (LoM) production plan.

Table 16-7 details the phase inventory that formed the basis of the SRK production schedule.

Table 16-7: Phase Inventory

Pit and Phase	Au Grade (g/t)	Ore Tonnes (Mt)	Waste Tonnes (Mt)	Total Tonnes (Mt)	Strip Ratio (W:O)	Contained Gold (koz)
PORTAL_P1		-	6.38	6.38		-
SNAKE_P1	2.52	3.33	21.43	24.76	6.44	269.2
SNAKE_P2	1.92	6.95	72.27	79.22	10.40	428.2
MILLZONE_P1	2.25	4.11	10.79	14.90	2.62	297.7
MILLZONE_P2	1.12	5.01	65.20	70.21	13.01	180.6
CHAMPION_P1	0.85	2.44	9.59	12.04	3.93	66.8
CHAMPION_P2	0.94	0.13	0.66	0.79	5.04	3.9
HAILE_P1	1.38	4.57	9.19	13.76	2.01	202.0
LEDBETTER_P1	2.47	1.75	54.95	56.69	31.47	138.8
LEDBETTER_P2	2.42	7.25	67.57	74.82	9.33	564.8
LEDBETTER_P3	1.14	4.43	62.93	67.35	14.22	162.8
LEDBETTER_P4	1.60	9.20	73.47	82.66	7.99	472.1
REDHILL_P1	1.22	5.88	26.76	32.64	4.55	231.3
Total	**1.71**	**55.04**	**481.17**	**536.21**	**8.74**	**3,018.2**

Source: SRK, 2017

<u>Production Schedule Targets</u>

The LoM production schedule was created in the Chronos scheduling package with annual ore and ounce targets defined by OceanaGold and SRK with the inclusion of the Horseshoe underground production. Because Haile is in operation and the LoM plan calls for a mill expansion and underground mine, the base assumptions for the ramp-up period are detailed in Table 16-8.

Table 16-8: Mine Plan Ramp Up

Year	Processing Rate Mtpa	Open Pit Ore Contribution (Mtpa)	Open Pit Total Material Movement (Mtpa)	Underground Ore Contribution (Mtpa)
2017	1.9	1.9	19.8	-
2018	2.5	2.5	22.3	-
2019	3.0	3.0	44.0	-
2020	3.3	3.3	44.0	-
2021	4.0	3.3	40.0	0.7

Source: OceanaGold

The SRK production schedule was based on January 1, 2017 pit topography and comprised of two years of monthly periods, two years of quarterly periods and 12 annual periods. SRK used the IBM ILOG CPLEX Optimizer (CPLEX) linear solver to meet the minimum and maximum tolerances detailed in Table 16-9 within precedence rules for each phase and bench. Bench sinking rates maxed at 1.5 per month except in the final year of production when the Champion pit was accelerated into a single year.

Table 16-9: Optimization Targets Excluding Bench Sinking Rate Limits

Period	Period Ounces		Ore Tonnes (Mt)		Total Tonnes (Mt)	
	Min	Max	Min	Max	Min	Max
2017	5,200	15,833	0.07	0.07	1.51	1.69
2017	12,500	15,833	0.13	0.15	1.71	1.80
2017	12,500	15,833	0.14	0.16	1.53	1.61
2017	12,500	15,833	0.15	0.17	1.37	1.44
2017	12,500	15,833	0.12	0.25	1.62	1.80
2017	12,500	15,833	0.02	0.25	1.67	1.80
2017	12,500	15,833	0.02	0.25	1.63	1.80
2017	12,500	15,833	0.17	0.25	1.67	1.80
2017	12,500	15,833	0.17	0.25	1.71	1.80
2017	12,500	15,833	0.17	0.25	1.71	1.80
2017	12,500	15,833	0.17	0.25	1.71	1.80
2017	15,833	16,667	0.17	0.25	1.71	1.80
2018	15,833	16,667	0.17	0.25	1.71	1.80
2018	15,833	16,667	0.17	0.25	1.71	1.80
2018	15,833	16,667	0.17	0.25	1.71	1.80
2018	15,833	16,667	0.17	0.25	1.71	1.80
2018	15,833	16,667	0.17	0.25	1.90	2.00
2018	15,833	16,667	0.17	0.25	1.90	2.00
2018	15,833	16,667	0.17	0.25	1.90	2.00
2018	15,260	16,667	0.17	0.25	1.90	2.00
2018	14,780	16,667	0.17	0.25	1.90	2.00
2018		16,667	0.17	0.25	1.90	2.00
2018		16,667	0.17	0.25	1.90	2.00
2018	15,833	16,667	0.17	0.26	1.90	2.00
2019	47,500	50,000	0.71	0.75	11.00	20.00
2019	47,500	50,000	0.71	0.75	11.00	20.00
2019	47,500	50,000	0.77	0.81	11.00	20.00
2019	47,500	50,000	0.71	0.75	11.00	20.00
2020	45,000	50,000	0.77	0.81	11.00	20.00
2020	47,500	50,000	0.77	0.81	11.00	20.00
2020	47,500	50,000	0.77	0.81	11.00	20.00
2020	47,500	50,000	0.77	0.81	11.00	20.00
2021			3.30	3.30	40.00	55.00
2022			3.30	3.30	39.00	55.00
2023			3.30	3.30	32.70	55.00
2024			3.30	3.30	40.00	55.00
2025			3.74	3.74	40.00	55.00
2026			4.00	4.00	40.00	55.00
2027			4.00	4.00	40.00	50.00
2028			4.00	4.00	40.00	50.00
2029			4.00	4.00	37.00	41.00
2030			4.00	4.00	27.00	100.00
2031			4.00	4.00	-	100.00
2032			-	4.00	-	50.00

Source: SRK, 2017

Production Schedule Results

The results of the SRK production schedule are detailed in Table 16-10 below.

Points of note on the schedule:

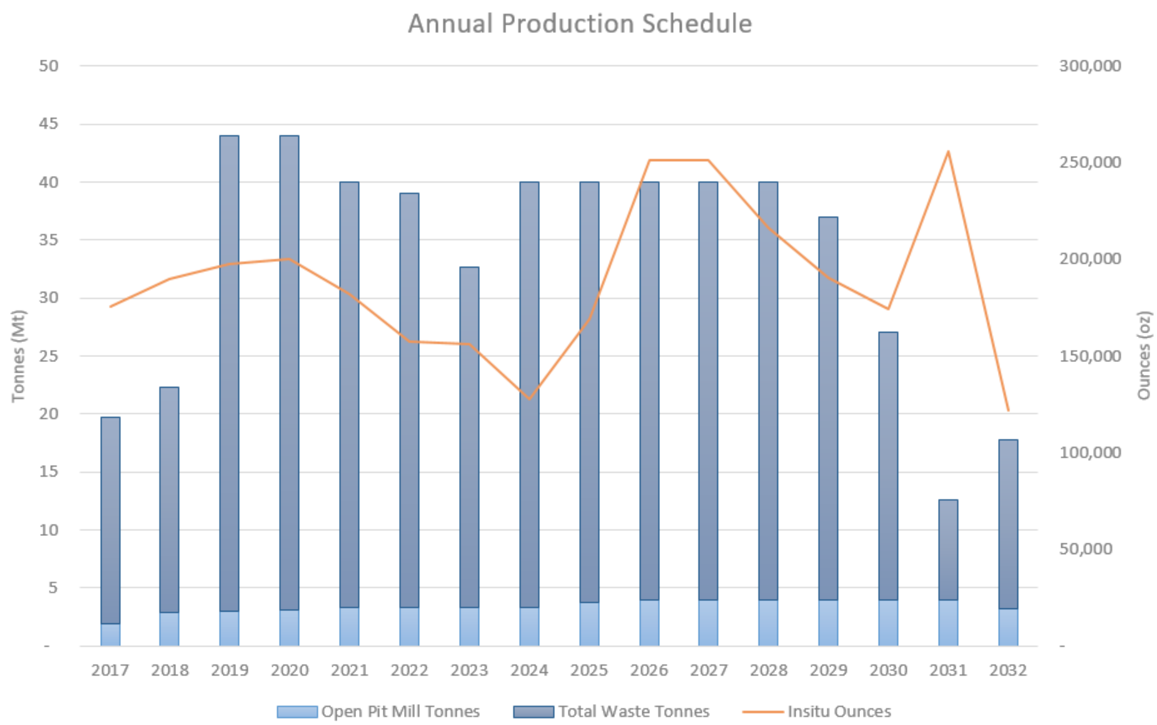
- The 2017 targets approximate current production and site budget targets for the year;
- 2018 total tonnage was increased to account for the portal cut excavation;
- Stripping was brought forward in 2019 and 2020 to keep ounce production above 175 koz/yr;
- Grade reduces after 2021 as the Haile and Red Hill phases are mined and Ledbetter ore is accessed; and
- Grade spikes in 2026 as high grade ore in Ledbetter is exposed.

Table 16-10: Open Pit Production Schedule

Year	Open Pit Mill Tonnes (Mt)	Total Waste Tonnes (Mt)	Total Tonnes (Mt)	Strip Ratio (W:O)	Gold Grams (g)	Gold Grade (g/t)	Gold Recovery	Recovered Gold (koz)	In situ Gold (koz)
2017	1.92	17.84	19.76	9.29	5,463,211	2.84	0.85	150.0	175.6
2018	2.86	19.43	22.30	6.79	5,913,835	2.07	0.84	158.8	190.1
2019	2.98	41.02	44.00	13.74	6,148,709	2.06	0.84	165.1	197.7
2020	3.13	40.87	44.00	13.08	6,220,700	1.99	0.83	166.6	200.0
2021	3.30	36.70	40.00	11.13	5,656,787	1.71	0.82	149.8	181.9
2022	3.30	35.70	39.00	10.82	4,905,153	1.49	0.81	128.4	157.7
2023	3.30	29.40	32.70	8.91	4,862,240	1.47	0.81	127.1	156.3
2024	3.30	36.70	40.00	11.13	3,966,617	1.20	0.80	101.9	127.5
2025	3.74	36.26	40.00	9.71	5,267,212	1.41	0.81	137.3	169.3
2026	4.00	36.00	40.00	9.00	7,820,260	1.96	0.83	209.2	251.4
2027	4.00	36.00	40.00	9.00	7,813,751	1.95	0.83	209.0	251.2
2028	4.00	36.00	40.00	9.00	6,736,256	1.68	0.82	178.1	216.6
2029	4.00	33.00	37.00	8.25	5,917,892	1.48	0.81	154.8	190.3
2030	4.00	23.00	27.00	5.75	5,427,136	1.36	0.81	140.9	174.5
2031	4.00	8.64	12.64	2.16	7,961,701	1.99	0.83	213.2	256.0
2032	3.22	14.60	17.82	4.54	3,793,881	1.18	0.80	97.3	122.0
Grand Total	**55.04**	**481.17**	**536.21**	**8.74**	**93,875,341**	**1.71**	**0.82**	**2,485.0**	**3,018.2**

Source SRK 2017

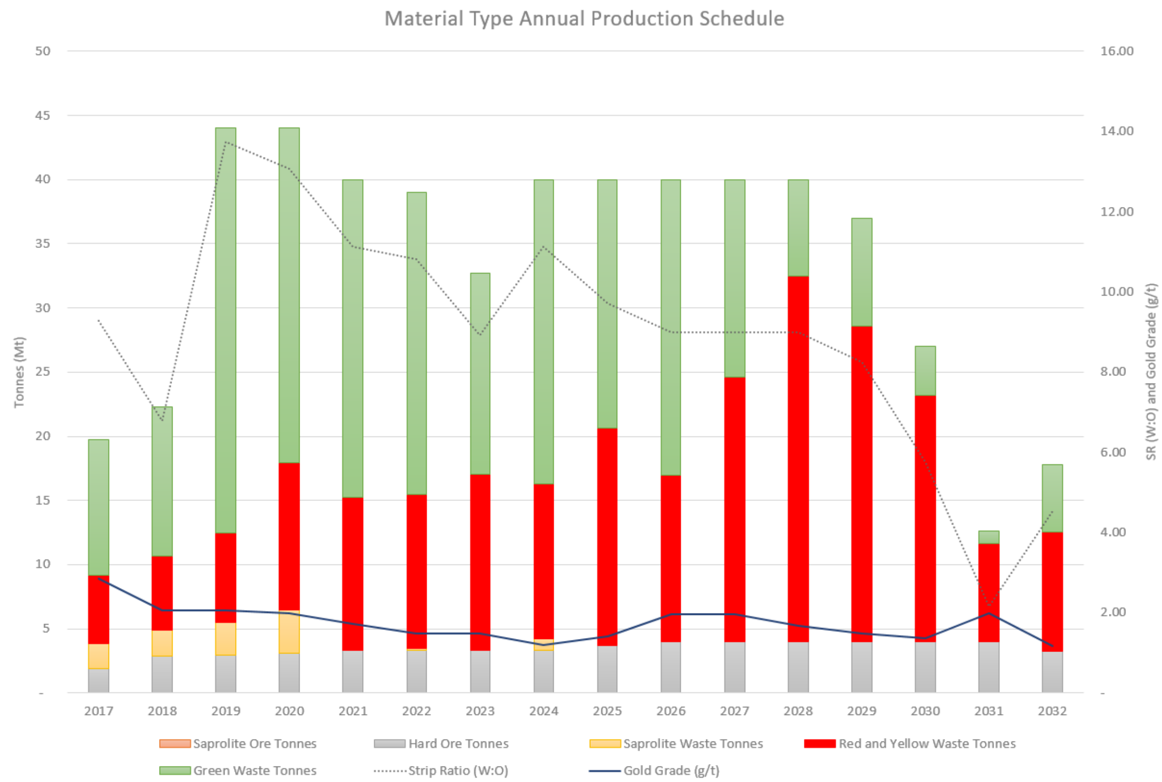
Figure 16-7 illustrates the annual production schedule for ore, waste and in situ ounces.



Source: SRK, 2017

Figure 16-7: Annual Production Schedule

Figure 16-8 breaks down the Red/Yellow waste, Green waste, Saprolite and Ore tonnes for the LoM. The Green waste is common early in the mine life but as the Ledbetter pit is exposed the amount of Red/Yellow waste increases.



Source: SRK, 2017

Figure 16-8: Material Type Annual Schedule

<u>Bench Sinking Rate</u>

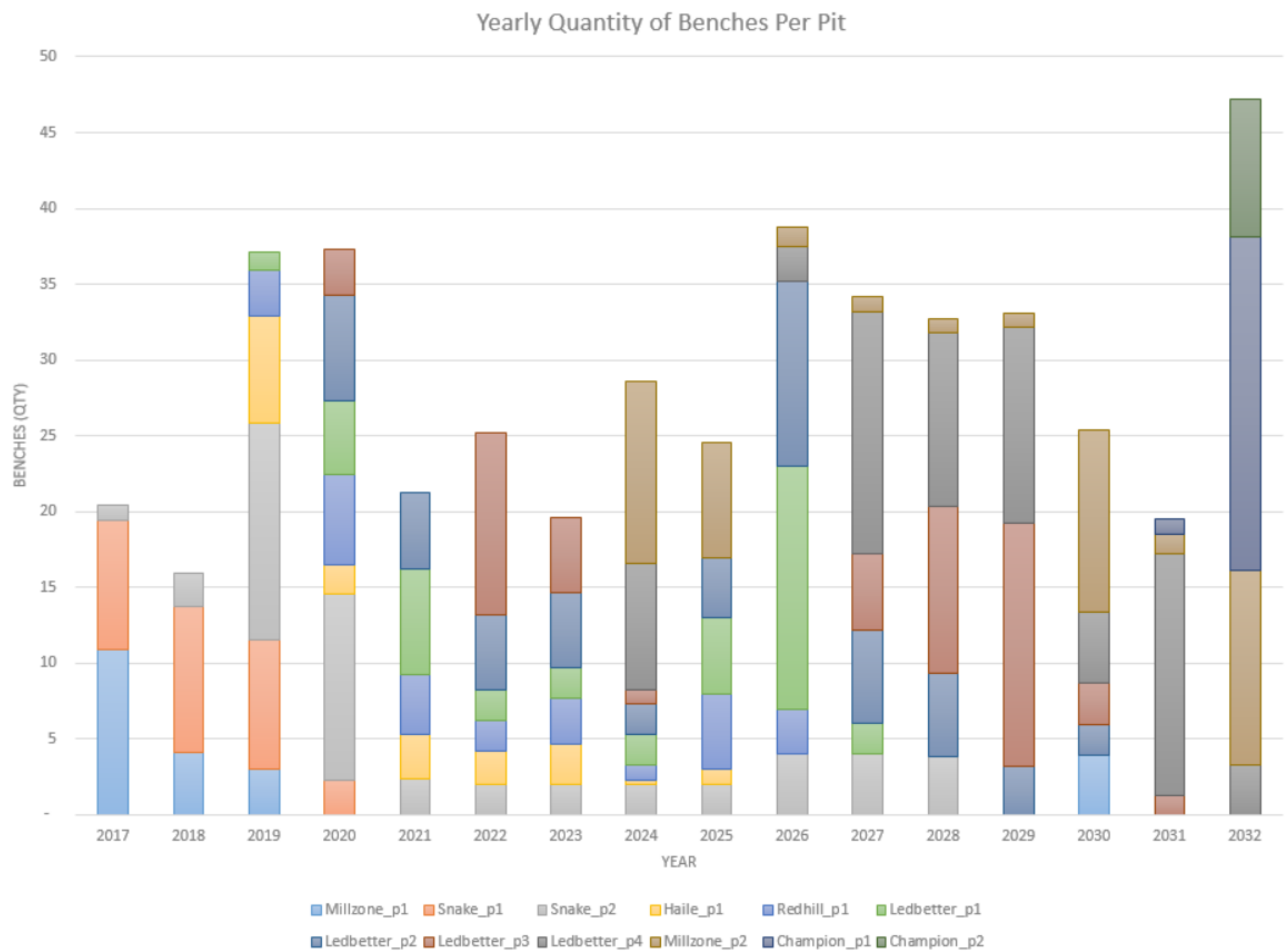
Generally, the number of benches mined within a pit phase within a given year fall below the one bench per month target bench sinking rate. The exceptions are in the Ledbetter pits which is essentially one large pushback broken into four parts. The mining is relatively straightforward when bench sinking rate is pressured. The Champion bench sinking rate was relaxed as it is the final pit to be mined in the last year of the schedule. Table 16-11 and Figure 16-9 details the annual benches mined from each pit/phase.

Figure 16-9 shows the benches mined from each pit/phase on an annual basis.

Table 16-11: LoM Yearly Bench Sinking Rates

Benches	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	Grand Total
Millzone_p1_Benches	11	4	3	-	-	-	-	-	-	-	-	-	-	4	-	-	22
Snake_p1_Benches	9	10	9	2	-	-	-	-	-	-	-	-	-	-	-	-	29
Snake_p2_Benches	1	2	14	12	2	2	2	2	2	4	4	4	-	0	-	-	52
Haile_p1_Benches	-	-	7	2	3	2	3	0	1	-	-	-	-	-	-	-	18
Redhill_p1_Benches	-	-	3	6	4	2	3	1	5	3	-	-	-	-	-	-	27
Ledbetter_p1_Benches	-	-	1	5	7	2	2	2	5	16	2	-	-	-	-	-	42
Ledbetter_p2_Benches	-	-	-	7	5	5	5	2	4	12	6	5	3	2	-	-	57
Ledbetter_p3_Benches	-	-	-	3	-	12	5	1	-	-	5	11	16	3	1	-	57
Ledbetter_p4_Benches	-	-	-	-	-	-	-	8	-	2	16	11	13	5	16	3	75
Millzone_p2_Benches	-	-	-	-	-	-	-	12	8	1	1	1	1	12	1	13	50
Champion_p1_Benches	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1	22	23
Champion_p2_Benches	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	9	9

Source SRK 2017



Source: SRK 2017

Figure 16-9: Benches Mined per Year

Haulage cycle times

Haulage is based on material from each phase going to either the green stockpiles or red/yellow stockpile or crusher location Because there are multiple ramps within the pit, the haulage profile within the pit to an exit point was controlled by a pit ramp haulage route. The in-pit haulage time is then added to the cycle time defined by haul road strings to the waste dumps or crusher from the pit exit. The cycle time and distance is stored in the block model and the haul cycles are matched with a routing block code in the block model. For example, waste blocks are assigned the waste cycle time based on the distance of the block to the waste dump. Strings to dumps are based on centroid locations. In the years where bulk fill is required for the tailings dam construction, the waste haul routes were extended to account for the additional haul times that will deliver waste rock at the required rate. Individual locations were not specified, but the nature of the tailings dam design means elevation changes will be minimal.

Haulage times reported in the block model are instantaneous times and do not have any efficiency corrections (other than speed around corners) delays, spot times or dump times. These corrections are made in the OceanaGold truck fleet estimation spreadsheet that uses the cycle times reported as the basis for the estimate.

The rimpull curve estimated speeds used in the cycle time estimation are detailed in Table 16-12 below.

Table 16-12: Rimpull Curve Representing Truck Speeds by Gradient

91-t Truck	Gradient (%)	Speed Uphill (km/h)	Speed Downhill (km/h)
Loaded	Flat	30	40
	0	30.0	40.0
	2	18.5	30.0
	4	17.0	25.0
	6	14.9	21.0
	8	11.5	19.0
	10	10.4	17.0
	15	5.0	5.0
Empty	Flat	30	40
	0	30.0	40.0
	2	30.0	30.0
	4	27.0	30.0
	6	23.0	24.0
	8	20.0	18.5
	10	17.5	15.0
	15	10.0	10.0

Source: SRK, 2016

16.1.7 Mining Fleet and Requirements

The Haile open pit equipment currently being used on site has been kept for ore mining (Cat 6020B excavator and Cat 777 haul trucks) on 5 m benches, with the larger Cat 6030 excavators and 785 haul trucks being scheduled for the higher material movement with the ramp up in mining movements from 2019. A Cat 993 FEL or similar sized machine will act as backup to the hydraulic excavators. Table 16-13 shows the estimation of the scheduled hours per year. Table 16-14 shows the assumed Mechanical Availabilities and Use of Availabilities for the equipment to estimate the potential operation hours per year for different equipment types. They give close to 6,000 operating hours per year for the primary digging units which is world class, but considered achievable for Haile.

Table 16-13: Estimation of Scheduled Hours per Year

Maximum Days Per Year	**365**
Holidays (Days)	0
Weekends (Days)	0
Possible Days/year	365
Weather & other lost production days	5
Operating Days/year	360
Scheduled hours/shift	12
Shifts/day	2
Scheduled hours/year	8,640

Source: OceanaGold

Table 16-14: Factors in Estimation of Potential Operating Hours

	Truck	**Excavator**	**FEL**
Mechanical Availability	0.82	0.82	0.85
Use of Availability	0.85	0.85	0.60
Utilisation	0.70	0.70	0.51
Operating Hours per Year	6,022	6,022	4,406

Source: OceanaGold

Waste drilling is based on using a Cat MD6290 with a 171 mm hole for a 10 m bench. For ore, a 5 m bench has been assumed using a 115 mm diameter hole. Cat 5150C drills are used for the ore, wall control and sampling. Penetration rates vary depending on rock type and range between 22 and 40 m/hr.

Four to five passes of the primary digging units will be used to load the matching trucks. Annual productivity rates have been estimated from equipment specifications, material characteristics, spot and loading times, truck presentation and primary digging unit propel factors, scheduled hours per year, mechanical availability and use of availability.

For estimating truck numbers, haul profiles were used to estimate annual average travel times by different material movements. For the LOM schedules the material movements were divided into:

- Green waste
- Red waste
- Yellow waste
- Tailing waste
- Saprolite waste
- Ore
- Rehandle underground ore
- Saprolite ore

A separate travel time was estimated for each material type and truck hours allocated to it. Table 16-15 shows the methodology for estimating truck requirements using the Cat 6030 and Cat 785 combination for the Green Waste mined in 2020.

Table 16-15: Trucking for Cat 6030 with Cat 785

Green Waste in 2020 (Mt)	26.02
Travel Time Green Wste (min)	19.34
Waste Payload (t)	134
Turn and dump (min)	1.00
Spot and manoeuvre (min)	0.50
Load truck (min)	1.87
Waste cycle time (min)	21.71
Efficiency (min/h)	55
Queue factor	0.95
Waste Productivity (t/h)	323
Mechanical Availability (%)	0.82
Use of Availability (%)	0.85
Waste Truck - Operating Hours	80,564
Waste Trucks Required	13.4

Source: OceanaGold

Table 16-17 lists the major production equipment required, while Table 16-20 shows additional equipment required. A down the hole service will continue to be provided by an explosives supplier.

Table 16-16: Major Equipment Numbers Required to Meet Mine Schedule

Fleet	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Cat 6020	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cat 6030	-	-	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Hit 1900 / Cat993	2	2	2	1	1	-	-	1	1	1	1	1	-	-	-	-
Trucks 777 - Ore	2	3	3	3	3	3	4	3	4	4	4	5	5	6	6	4
Trucks 785 - Waste	9	8	18	22	18	15	21	21	27	18	26	27	23	22	10	13
Ore Drill - Cat 5150C	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Waste Drill - Cat MD6290	2	2	4	4	4	4	3	4	4	4	4	4	3	3	1	2

Source: OceanaGold

Table 16-17: Open Pit Fleet Additions

Fleet	Existing	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	Total Purchased	Total LoM
Cat 6020	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1
Cat 6030		-	-	2	-	-	-	-	-	-	-	1	-	-	-	-	-	3	3
Hit 1900 / Cat993	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2
Trucks 777 - Ore	12	-	-	-	-	-	-	1	-	1	-	1	-	1	1	-	-	5	17
Trucks 785 - Waste	3	-	-	16	3	-	-	-	-	6	-	-	-	3	3	1	-	32	35
Ore Drill - Cat 5150C	2	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	1	3
Waste Drill - Cat MD6290	3	-	-	1	-	-	1	-	1	-	1	1	-	-	-	-	-	5	8

Source: OceanaGold

Ancillary equipment to support the load and haul and drilling fleets currently includes four tracked and one rubber tyred dozer. This will be increased with an extra dozer starting in 2019. In addition, two graders cover the pit, dump and surface roads. A FEL provides stockpile and extra loading capability. Two water trucks are assigned to roads and servicing drills. Other equipment includes lighting plants, sumps pumps, fuel truck compactor and light vehicles.

16.1.8 Labor

Labor costs on an annual basis have been built up with reference to equipment unit numbers, the roster and requirements for management and technical staff. The roster is a continuous three panel crew roster of 12-hour shifts with two shifts per day. Rostered days off, leave, training, sick leave and absenteeism allowances results in 217 days worked per year. Operator numbers are based on allocating specific numbers to each equipment item (generally three people on the basis of the three-panel roster). Spare personnel are added at the rate of 10% to allow for vacation, sickness, absenteeism and training. The manpower numbers assume an efficient workforce operating with high level of multi-skilling and flexibility. The tradesmen have been estimated using a ratio of maintenance hours/unit operating hour as given in Table 16-18 below. An allowance of 10% has been added to account for small equipment maintenance and facilities. Table 16-19 shows the required workforce for the year 2020.

Table 16-18: Trade Ratios

Unit	Trade hours / Equipment operating hours
Excavators	0.66
Trucks	0.44
Dozers Tracked	0.40
Rubber Tired Dozer	0.40
FEL	0.45
Grader	0.30
Drill	0.55
Water Truck	0.44
Other	0.30

Source: OceanaGold

Table 16-19: Labor Levels in 2020

Labor Category	Year 2020
Management and Technical Services	
Mines Operations Management	7
Mine Engineering/Geology/Survey	20
Mine Projects/Tech Services	11
Total	**38**
Operations Labor	
Mine Foreman/Supervisors	5
Operators	127
Blast Crew/ Explosives Ops	0
Holiday/Training/Sickness, etc	12
Total	144
Maintenance Labor	
Maintenance Super/Shift Foreman	5
Fitter/Welder/Electrician/Serviceman	60
Maintenance Planner	1
Maintenance Clerk	1
Holiday/Training/Sickness, etc.	6
Total	**73**
Grand Total	**255**

Source: OceanaGold

The manpower numbers are then applied against the annual labor cost by job type. These costs have been supplied by Haile operations and are inclusive of on-costs.

16.1.9 Mine Dewatering

The relevant stratigraphy of the Project area includes (from oldest to youngest) basement rocks that have been intruded by mafic and diabase dikes, saprock (basement rock that is weathered along failure zones), saprolitic clays (strongly weathered basement) and Coastal Plain Sand (CPS).

The project area is located within the Carolina Slate Belt along the contact area between the Proterozoic-Cambrian metamorphosed volcanoclastic and metamorphosed sedimentary rocks that have been intruded by granites during the Permian-Carboniferous periods. Doloritic intrusions (dikes) also occurred during the Mesozoic in a linear northwest to southeast orientation in the project area. Contact metamorphic aureoles are variably associated with the intrusive rocks which have likely given rise to significant shallow fracturing observed in project area basement rocks and enhanced secondary permeability, particularly in the metavolcanics and coarser grained metasediments.

The rocks of the Project area have been subject to deep erosion and extensive top-down weathering, largely due to long-term humid climatic conditions. This weathering has altered, to a varying extent, the original composition and texture of basement rocks, giving rise to the saprock and saprolitic clays that are seen today overlying more competent basement rocks. The secondary structural fabric of the basement and intrusive rocks, though, appears to be maintained to some extent within the weathered (saprolite and saprock) zone, although structure-related secondary porosity may not remain.

The CPS is inferred to thin to the west of the Project area, and comprises three separate sequences – an upper layer composed of poorly sorted quartz sand, a middle layer composed of quartz sand and clay / silt, and a lower lateritic layer composed of coarse gravels and sand. Along the major drainages of the site, e.g., HGMC, the CPS is largely absent, having been removed by erosion.

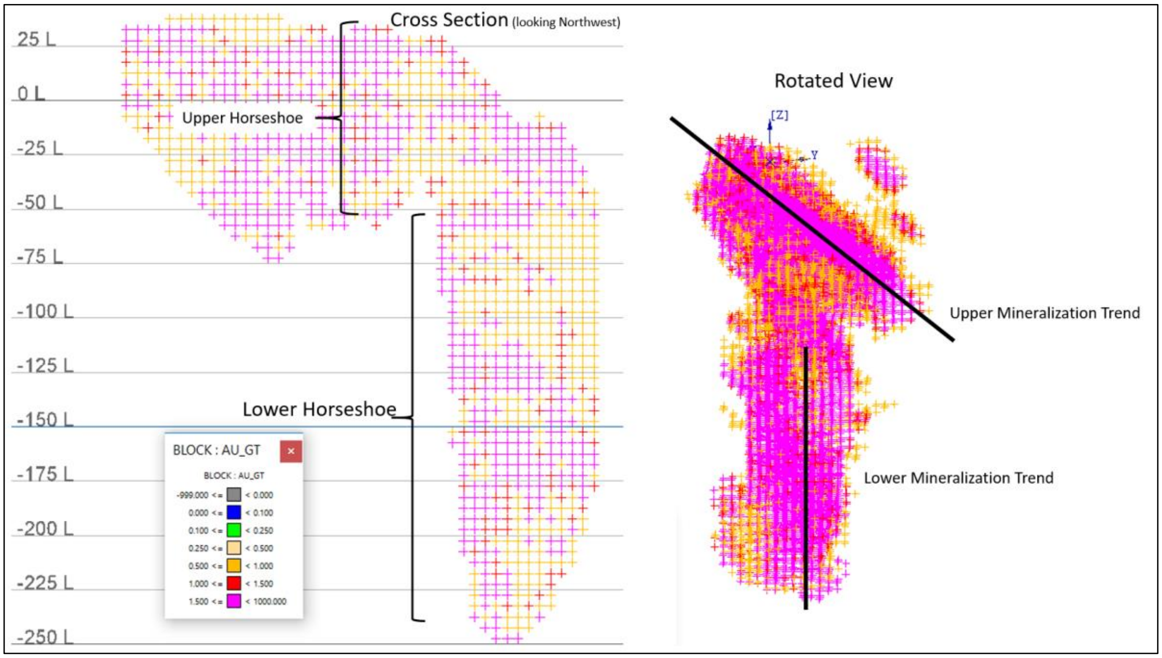
Based on monitoring of open pit mining operations data around Mill Zone and Snake Pit, most of the produced groundwater from dewatering is occurring through the fractured bedrock stratigraphic unit, which is approximately 50 m thick and the top of this unit occurs at depths ranging from 30 to 40 m below land surface. Although there is groundwater level drawdown from dewatering in the overlying saprolite/saprock and CPS there is very little flow contribution from these stratigraphic units. There is limited groundwater flow from the competent bedrock since it is hydraulically connected to the fractured bedrock layer but the flow is primarily through the matrix and the hydraulic conductivity of the competent bedrock is very low.

A numerical groundwater flow model was developed for the project using the MODFLOW-SURFACT code to represent the stratigraphy above as 5 model layers: 1) Layer 1 – CPS, 2) Layer 2 – Saprolite/Saprock, 3) Layer 3 – Fractured Bedrock, 4) Layer 4 – Competent Bedrock/Metavolcanics, and 5) Layer 5 – Competent Bedrock/Metasediments. In plan view, the model covers an area of 900 km^2 with the Haile site roughly in the middle of the model grid. Site stratigraphic data, meteorological data and groundwater pressure/level data from numerous monitoring wells and vibrating wire piezometers installed in each of the stratigraphic units, and groundwater production data were used to develop the conceptual model and calibrate the numerical model.

After the model was calibrated under both steady-state and transient conditions it was used to simulate quantities and rates of dewatering required to achieve and control water to allow safe access to the open pits and underground during operations as mining progresses both horizontally and vertically with time. A combination of perimeter (ex-pit) and in-pit pumping wells and sump dewatering from the active mine areas will be required to manage groundwater pressures so they don't impact on mine production. Simulations of future dewatering of the open pit areas and underground completed to date indicate dewatering rates will range from 7,500 m^3 per day to 11,000 m^3 per day and average around 9,500 m^3 per day. The timing and volume of extracted groundwater is expected to be manageable

16.2 Underground Mining Methods

The Project is currently being mined as an open pit mine; however, economic mineralization extends below and outside of the pit extents. This mineralization, not mined by the current feasibility study open pit described in this document, is assessed herein as an underground mine and is referred to as Horseshoe. The upper portion of the Horseshoe mineralization is wider and appears to be localized in planar zones along foliation and also in deformed vein style zones of silicification. This zone also strikes NE-SW, but dips at 40º to 45º NW. The lower zone is clearly controlled by lithology where the gold mineralization is localized in deformed vein style zones of silicification within the metasediment parallel to the metavolcanic contact. This zone strikes NE-SW and dips vertically. Figure 16-10 shows the Horseshoe upper and lower mineralized zones.



Source: SRK

Figure 16-10: Horseshoe Upper and Lower Mineralized Zones

Based on the orientation, depth, and geotechnical characteristics of the mineralization, a transverse sublevel open stoping method (longhole) with has been selected. The stopes will be 15 m wide and stope length will vary based on mineralization grade and geotechnical considerations. A spacing of 25 m between levels is used. CRF will be used to backfill 75% of stopes and non-cemented waste rock will be used in the remaining stopes. The CRF will have sufficient strength to allow for mining adjacent to backfilled stopes.

16.2.1 Cut-off Grade Calculations

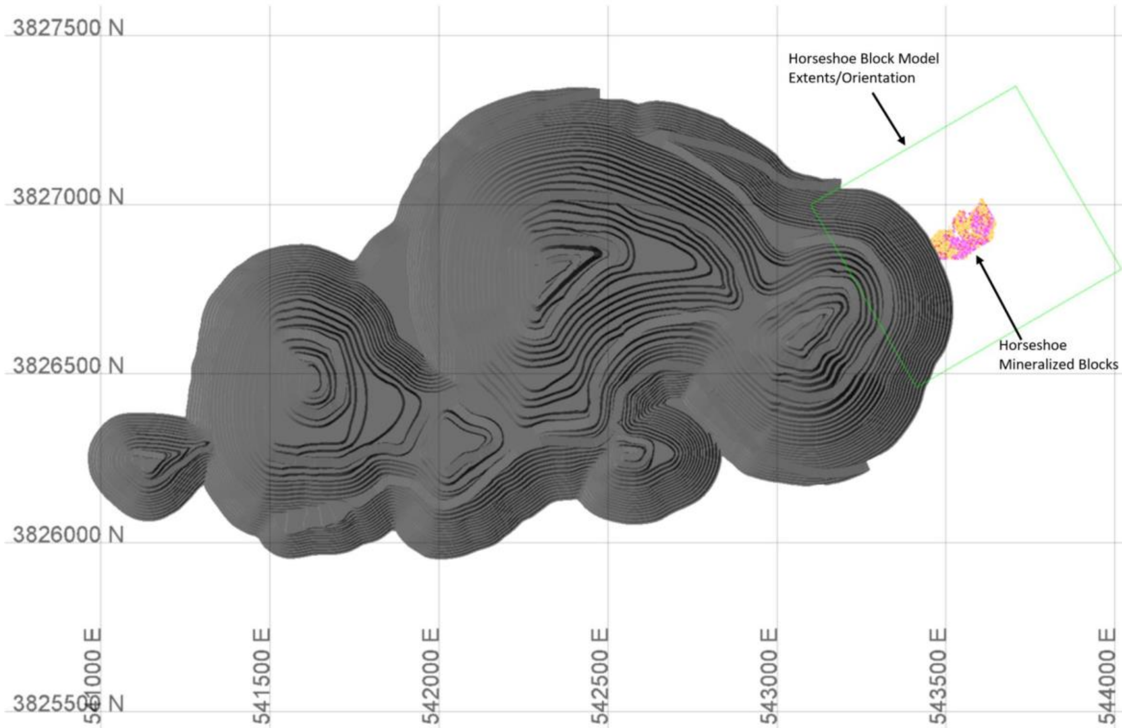
Current estimated project costs and the calculated Au CoG are shown in Table 16-20. For mine design purposes, a minimum cut-off of 1.5 g/t Au was used.

Table 16-20: Underground Cut-off Grade Calculation

Parameter	Amount	Unit
Mining cost (1)	42.00	US$/t
Process cost	10.50	US$/t
G&A	2.75	US$/t
Total Cost	**$55.25**	**US$/t**
Gold price	1,300.00	US$/oz
Average Au mill Recovery (2)	88%	
Smelting & Refining	1.28	US$/oz
CoG	1.50	g/t

Source: SRK, OceanaGold
(1) Includes backfill
(2) Average stated. Variable recovery is expected based on head grade based on the following equation: (1-(0.2152*au grade^-0.3696))

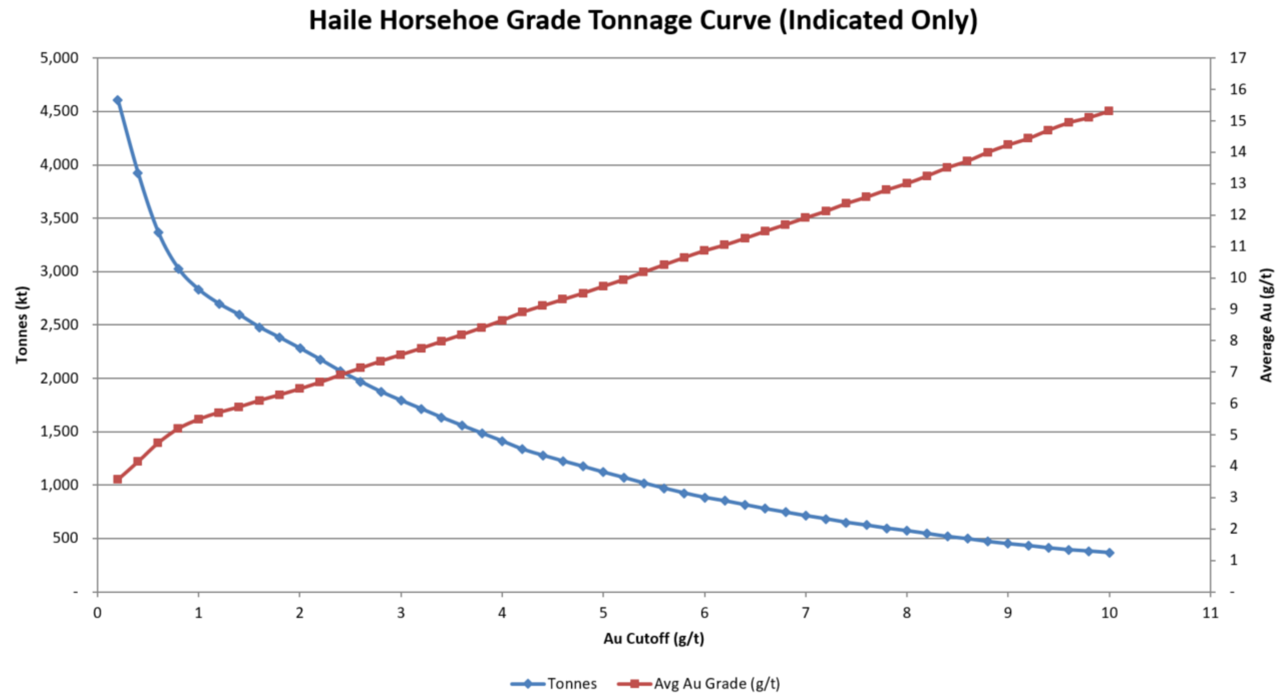
The basis for the FS mine design work is the underground resource model described in Section 15.2. The model is rotated to fit the general mineralization/foliation trends. Figure 16-11 shows the block model orientation and mineralized blocks.



Source: SRK

Figure 16-11: Haile Underground Block Model and Mineralization Extents

Figure 16-12 shows a grade-tonnage curve for the Horseshoe deposit. The underground Mineral Resources shown in Figure 16-3 are classified as Indicated.



Source: SRK

Figure 16-12: Horseshoe Underground Model Grade/Tonne Curve based on Au Cut-off

16.2.2 Geotechnical

The 2017 geotechnical field investigation consisted of 11 drill holes designed to examine rock mass fabric and structural features in and around the mineralized zone at different depths and orientations. Drill holes were drilled at varying orientations into the hangingwall, footwall, and mineralized rock. The field investigation included drilling of core, geophysical borehole logging of structural features, geotechnical core logging, core sample collection for laboratory strength testing. A total of 4,217 m of core was characterized. Two previous geotechnical characterization campaigns have been conducted (Golder, 2011; SRK, 2016).

Data from all three campaigns have been combined into a single database. Table 16-21 is a summary of the 7,345 m of core that has been logged for geotechnical characterization. The field program included in situ stress measurements. A summary of the laboratory testing program for rock strength is provided on Table 16-22.

Figure 16-4 shows a 3D perspective view of the underground workings with the as-drilled characterization core holes from each drilling campaign. The hole diameters have been enlarged to 40 m to simulate the representative coverage volume for each hole. This figure illustrates that most of the planned mining area and underground infrastructure has been covered by characterization.

Table 16-21: Discontinuity Orientation Data for 2014 Geotechnical Investigation

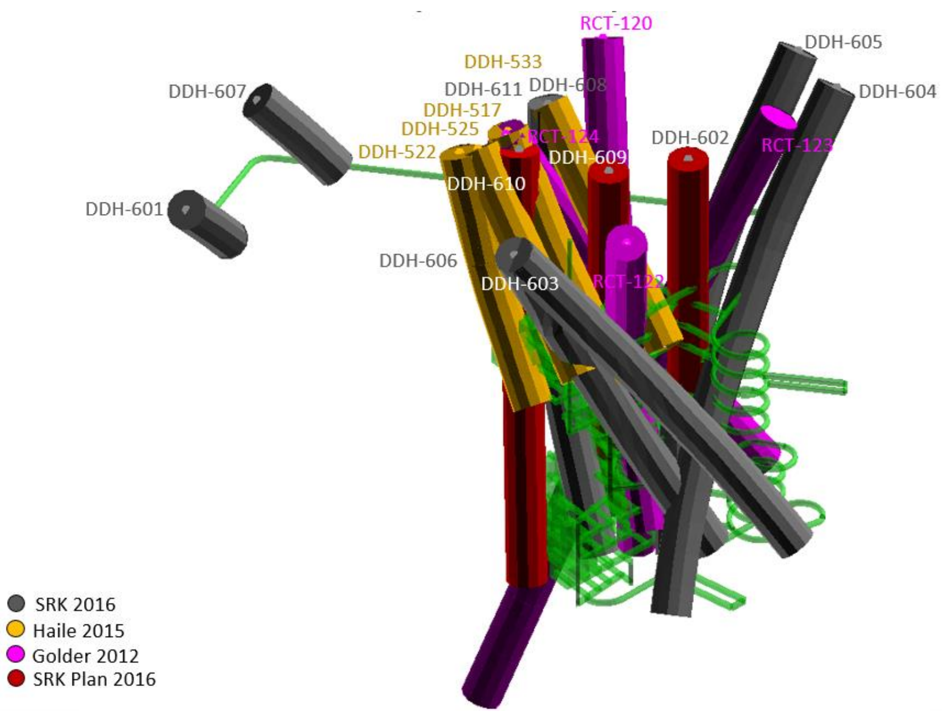
Drill hole ID	Drill hole Length (m)	Total Discontinuities		Total Discontinuities Logged
		Logged	ATV	
RCT-120	475.8	469	-	469
RCT-122	535.8	273	-	273
RCT-123	693.6	277	-	277
RCT-124	461.7	365	-	365
DDH-517	264.0	215	-	215
DDH-522	258.3	381	-	381
DDH-524	179.8	436	-	436
DDH-533	259.1	315	-	315
DDH-601	121.9	179	425	604
DDH-602	406.9	134	648	782
DDH-603	450.5	364	1,347	1,711
DDH-604	519.1	481	717	1,198
DDH-605	504.9	577	-	577
DDH-606	432.2	625	1,021	1,646
DDH-607	113.4	153	300	453
DDH-608	399.9	226	767	993
DDH-609	403.9	137	805	942
DDH-610	438.3	674	1,404	2,078
DDH-611	426.4	651	1,024	1,675
Total	**7,345.5**	**6,932**	**8,458**	**15,390**

Source: SRK, 2017

Table 16-22: Summary of the Laboratory Tests

Year	Laboratory	Rock Type	UCS	UCSM	TCS	BT	DSS	Total
2009 (Golder OP)	ATT	Metasediments	1	1	4		2	13
		Dike	1	1			-	
		Metavolcanics	2	1			-	
2011 (Golder UG)	ATT	Metasediments	12					12
		Dike						
		Metavolcanics						
2016 (SRK UG)	Agapito	Metasediments	7	8	5	1	3	37
		Dike	1	1			1	
		Metavolcanics	2	2	3	1	2	
2017 (SRK UG)	Agapito	Metasediments	11	9	10	10	6	125
		Dike	3	4		2	1	
		Metavolcanics	21	16	15	9	8	
Total			**61**	**43**	**37**	**23**	**23**	**187**

Source: SRK, 2017



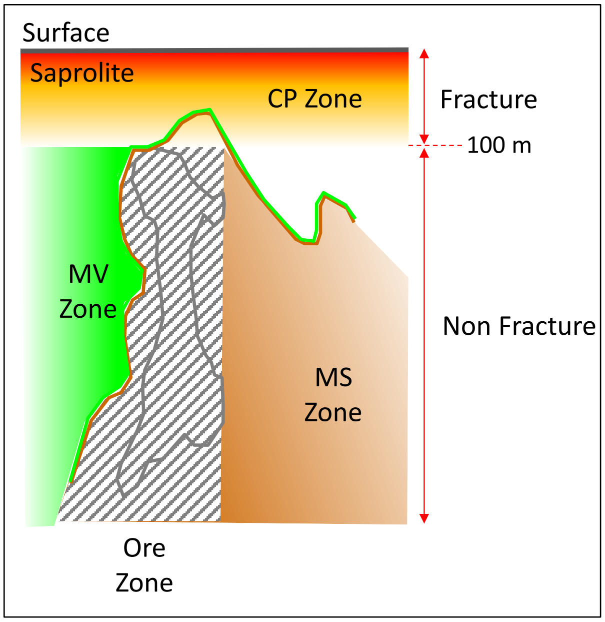
Source: SRK, 2017

Figure 16-13: Horseshoe Underground Workings with As-Drilled Characterization Core Holes

Rock Mass Characterization

Most of the rock quality designation (RQD) values indicate fair to good rock quality throughout the drill holes (i.e., RQD = 80 to 100). A total of 4,220.5 m of core length were logged with the majority being located within mineralized zone. Areas with lower RQD (i.e., RQD = 10 to 60) were mainly associated with weathered or altered rock and minor geological intrusions (e.g., weathered and fractured metasediments). Fracture frequency (FF) in the crown pillar ranged between 0 to 15 fractures per meter with an average of 6.5 fractures per meter. The fracture frequency in the ore body ranged between 0 to 8.3 fractures per meter with an average of 2 fractures per meter.

A northwest-southeast cross section through the geotechnical model is illustrated in Figure 16-14 as viewed looking towards the southwest. The crown pillar is the area above the mineralized zone in the fractured metasediments (MS) and metavolcanics (MV). The hangingwall is in the metasediments. The mineralized zone is mostly metasediments with some metavolcanics. The footwall is in the metavolcanics.



Source: SRK, 2017

Figure 16-14: Geotechnical Model Cross Section

The logged core was generally observed to be fresh to slightly weathered with weathering usually limited to the surfaces of the discontinuities of slight rock mass alteration. Field strength tests indicate that the rock was on average strong (R4). The rock would generally break along pre-existing planes of weakness such as veins, foliation and healed structural features.

Laboratory uniaxial compression strength test (UCS) results show that the strength of the weathered metavolcanic rocks are in the medium-strong range (UCS = 40 to 78 MPa), and fresh metavolcanic strength range is very strong (UCS = 110 to 195 MPa). The strength of the foliated metasediments range is medium strong (UCS = 32 to 70 MPa), and the non-foliated metasediments strength is considered very strong (UCS = 107 to 150 MPa). The dikes are very strong (UCS = 100 to 190 MPa). Table 16-23 through Table 16-25 provides a summary of the laboratory testing results.

Table 16-23: Summary of Point Load (Is_{50}) Test Results

Geotechnical Unit		Mine Zone	UCS Lab (MPa)	K Value	UCS PLT (MPa)			Field Estimated Strength Category (ISRM)
Lithology	Weathering				Mean	Lower Boundary (Px<20%)	Upper Boundary (Px<80%)	
Metavolcanics	Weathered	Crown Pillar	60	18	50	13	85	R2-R3
	Fresh	Hanging Wall Ore Body Footwall	153	27	106	70	145	R4-R5
Metasediments	Fresh	Hanging Wall Ore Body Footwall	105	21	70	34	105	R3-R4

Source: SRK, 2017

Table 16-24: Summary of Strength Properties (mi and σ_{ci})

Geotechnical Unit		Density (t/m^3)	Laboratory Test		Intact Rock Properties from TCS		Elastic Constants	
Lithology	Weathered		σ_t (MPa)	UCS (MPa)	σ_{ci} (MPa)	mi	Ei (GPa)	ν
Metavolcanics	Weathered	2.60	4.4	60	61	14	21.9	0.28
	Fresh	2.70	12.7	153	158	13	73.4	0.24
Metasediments Foliated	Fresh	2.76	6.2	50.5	59	10	41.6	0.24
Metasediments Not Foliated		2.75	11.8	127.8	138	12	58.9	0.24
Dike	Fresh	2.92	12.94	146	-	-	78.7	0.22

Source: SRK

Table 16-25: Summary of Elastic Properties (E_i and ν)

Geotechnical Unit		Mine Zone	UCS (MPa)			E_i (GPa)	ν
Lithology	Weathering		Mean	Low Boundary (Px<20%)	Upper Boundary (Px<80%)		
Metavolcanics	Weathered	Crown Pillar	57	35	80	25.8	0.21
	Fresh	Hanging Wall Ore Body Footwall	148	105	192	73.6	0.23
Metasediments	Fresh	Hanging Wall Ore Body Footwall	92	65	120	62.6	0.21
Dike	Fresh		146	100	190	78.7	0.22

Source: SRK, 2017

The Rock Mass Rating (RMR_{76}) values ranged from 49 to 68 in the hangingwall rock, with Barton Q' values varying from 4.3 to 19, with most of the rock mass being of fair to good quality. In the footwall metavolcanics RMR_{76} values were in the 64 to 88 range, with Q' values varying from 7.7 to 30, with most of the rock mass being of Good quality. The mineralized rock had the greatest RMR_{76} variations, with values between 56 and 79, while Q' values ranged from 5.8 to 24. Table 16-26 summarizes the rock mass characterization data. Overall, the rocks are Fair to Good quality. The table also indicates the quantity or core in each domain.

Table 16-26: Summary of Rock Mass Quality by Domain

Geotechnical Domain	Lithology	Density (t/m^3)	IRS (MPa)	RQD (%)	Fracture Frequency (FF/m)	RMR76/ GSI	Q'
Crown Pillar (13%)	MV	2.60	13 to 86	61 to 100	0 to 15.0	45 to 69	2.8 to 13.5
	MS	2.63	26 to 64	41 to 91	1.2 to 19.2	43 to 62	2.3 to 8.3
	DB	2.91	22 to 140	72 to 100	0 to 11.8	47 to 68	2.2 to 11
Hanging Wall (22%)	MV	2.69	41 to 219	88 to 100	0 to 3.7	63 to 82	10.7 to 26
	MS	2.76	22 to 97	70 to 100	0 to 10.6	49 to 68	4.3 to 19
	DB	2.96	42 to 173	45 to 100	0 to 26.6	39 to 72	0.9 to 9.6
Ore Body (50%)	MV	2.70	86 to 205	85 to 100	0 to 5.8	64 to 89	8.3 to 34
	MS	2.75	37 to 125	75 to 100	0 to 8.3	56 to 79	5.8 to 24
Footwall (15%)	MV	2.69	86 to 197	83 to 100	0 to 6.5	64 to 88	7.7 to 30
Decline	DCL <100 m MV	2.60	13 to 121	64 to 100	0 to 15.7	46 to 74	1.7 to 18
	DCL >100 m MV	2.70	33 to 159	69 to 100	0 to 9.8	52 to 83	5.2 to 27

Source: SRK, 2017

Mine Design Parameters

Transverse long-hole mining has been selected as the mining method for the entire orebody. This method requires upper and lower access drifts to be developed from the footwall haulages with a proposed 25 m vertical spacing between levels and a total open stope height of 30 m. The overall open stope dimensions are 15 m wide x 30 m high x 30 m long (maximum length).

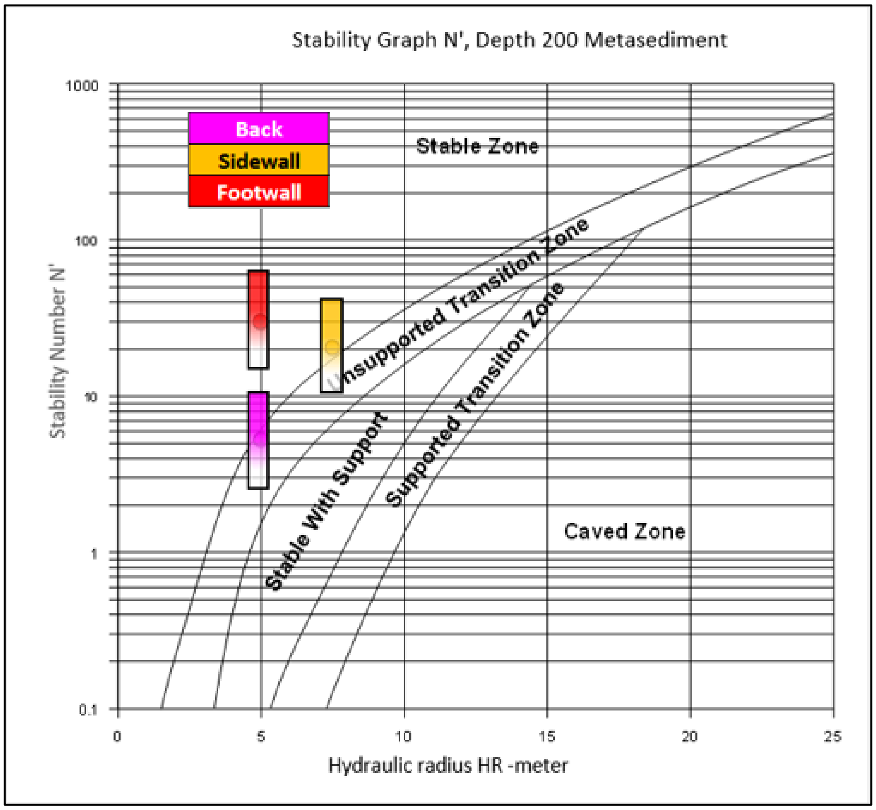
Empirical methods of stope design have been employed to evaluate stability conditions. The Stability Graph Method (Mathews et al., 1981), as modified by Potvin et al. (2001), is based on more than 480 case histories worldwide and has been used to size stopes so they remain stable during mining. The stability graph method plots the stability number (N') on the vertical axis against the hydraulic radius (wall area divided by wall perimeter) of the stope wall or back on the x axis. The stability number is calculated based on the rock mass quality (Q' system), geologic structure and induced stress conditions. The stability of the stope walls has been checked using Matthews method (Figure 16-15 and Figure 16-16) to ensure that remote mining equipment can extract all the ore in an unsupported stope without being buried by falling ground.

Stability of the stopes have been checked using a mine-scale numerical model. The FLAC3D program was used to simulate the mining sequence. Results of the analysis confirm that the stopes are predicted to remain stable during active mining.

To minimize mining-induced damage to long-term access drifts, the setback distances used in the design include:

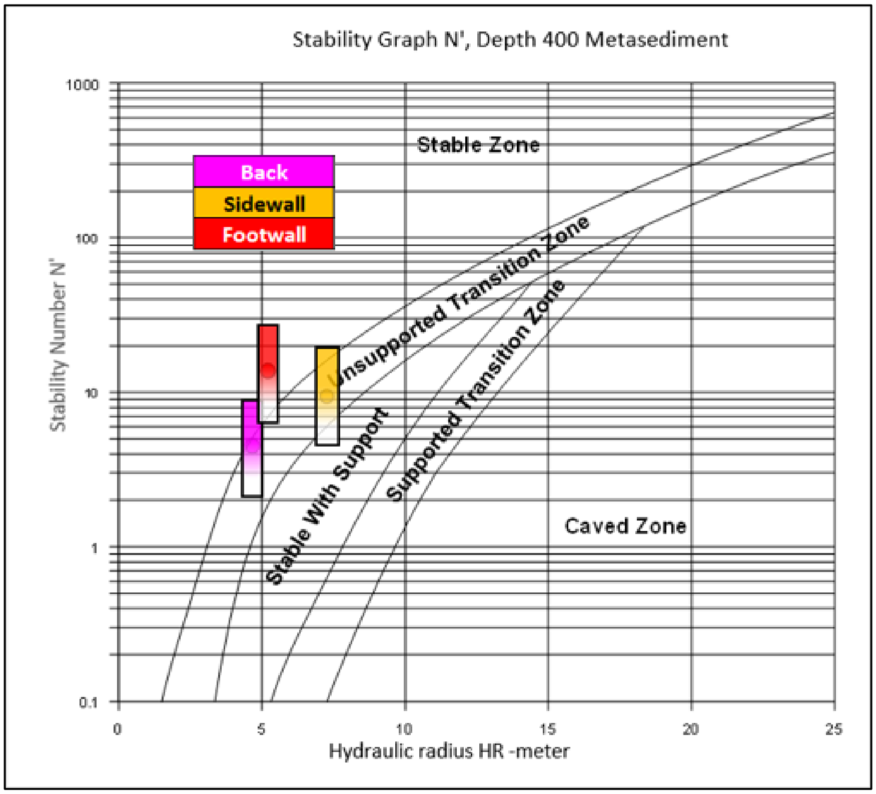
- Haulage setback of 18 m to 22 m from stopes; and
- Main ramp setback of 64 m to 70 m from stopes.

These values were also confirmed by the results from the numerical model by evaluating induced shear strain locations.



Source: SRK

Figure 16-15: Empirical Stope Design Chart (Potvin, 2001) Metasediments 200 m Depth



Source: SRK

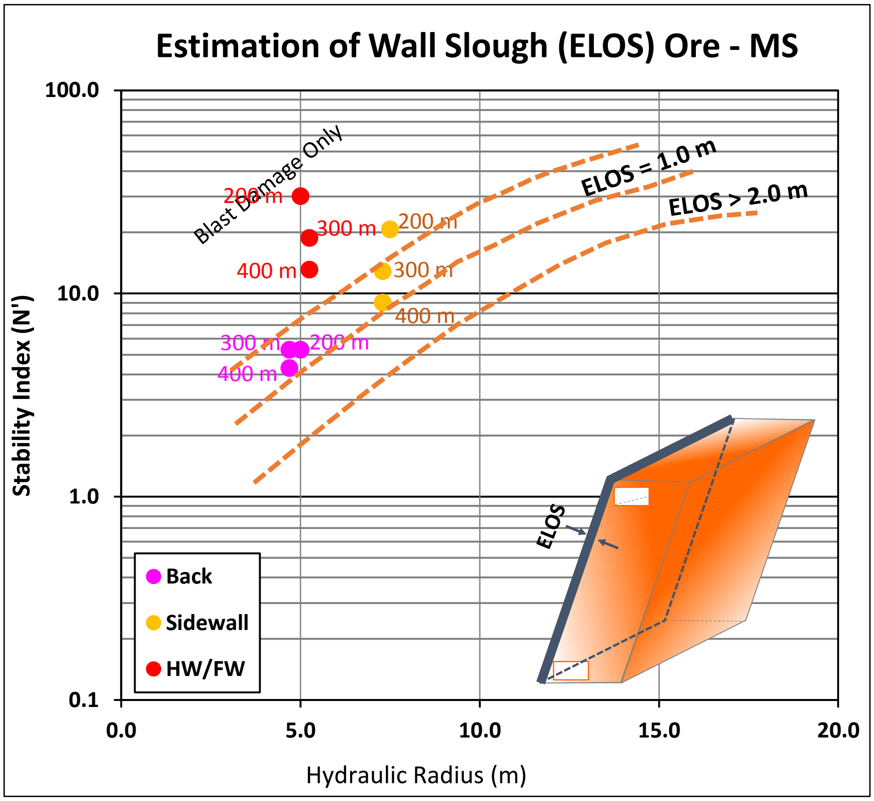
Figure 16-16: Empirical Stope Design Chart (Potvin, 2001) Metasediments 400 m Depth

Dilution was estimated using the method developed by Clark and Pakalnis (1997) based on an extensive set of case histories for open stopes. The method predicts the quantity of unstable wall rock for a given rock mass quality from a given stope size. The thickness of external dilution is estimated as equivalent linear overbreak/slough (ELOS). Table 16-27 summarizes the ELOS dilution assumptions used for mine planning purposes. Figure 16-17 shows the range of predicted ELOS in moderately weathered metasediments. Similar estimates were made for open stopes in other rock types and stress conditions.

Table 16-27: Summary of Dilution ELOS Estimates

Type	ELOS Value (m)
Sidewalls (rock)	0.50
Sidewalls (backfill)	0.25
Endwalls (rock & backfill)	0.15
Bottom (backfill)	0.05

Source: SRK



Source: SRK, 2017 adopted from Clark and Pakalnis, 1997

Figure 16-17: Estimation of Wall Slough (ELOS) NFZ-MS

Backfill

The required strength of cemented rockfill placed in primary stopes has been checked for stability under the critical conditions of mining the adjacent secondary stopes. The secondary stopes must remain stable so remote equipment can safely muck the ore without being buried by ground fall. A numerical model was used to assess the minimum backfill strength requirements. The model results indicate that a minimum UCS strength of 0.7 MPa is required after 14 days of curing to accommodate the mining cycle. Laboratory tests results on batches of cemented rockfill indicate that this required strength can be achieved using 4% cement by mass and a water-to-cement ratio of 2.0 with aggregate from waste rock currently available. Table 16-29 shows the preferred aggregate blend for use in backfill for both coarse fraction and fines fraction.

Table 16-28: Preferred Aggregate Blend

Size Description	(mm)	(Inch)	Percent Mass
% Boulder	256	10.1	0
% Large Cobble	128	5.0	3
% Cobble	64	2.5	4
% Coarse Gravel	32	1.3	8
% Medium Gravel	16	0.6	11
% Fine Gravel	8	0.31	15
% Coarse Pebble	4	0.16	16
% Fine Pebble	2	0.08	15
% Coarse sand	1	0.04	10
% Med sand	0.5	0.02	8
% Fine sand	0.25	0.01	6
% Silt/clay	0.125	0.00	4

Source: SRK

Ground Support and Stability

Ground support requirements were estimated using empirical support charts developed by Barton (1974). The method relates the rock mass quality (Q) to the equivalent dimension of the excavation (*De*). *De* is the ratio of the excavation width (D) to the excavation support ratio (ESR) index, which relates the use of the excavation to the degree of safety required. ESR parameters were assigned to each type of excavation within the framework of the empirical method. Table 16-29 is a summary of the drift dimensions for which ground support has been provided.

Table 16-29: Design Parameters for Different Drift Types

Excavation	Type Excavation	Opening Dimension W x H (m)	ESR	D	De
Decline <150 m depth	Long Term (LOM)	5.0 x 5.5	1.6	5.5	3.4
Main Ramp >150 m depth	Long Term (LOM)	5.0 x 5.5	1.6	5.5	3.4
Footwall Accesses	Medium Term (0.5 to 1 year)	5.0 x 5.0	2	5.0	2.5
Stope Accesses	Short Term (1 to 6 months)	5.0 x 5.0	2.5	5.0	2.0

Source: Barton, 1974

Various levels of support were specified for each range of rock quality. Table 16-30 is a summary of rock mass quality for each ground support type. The long-term ground support for access drifts in the footwall area have been specified to maintain safe access to infrastructure and ore haulage from the stopes back to the hoisting shaft while those drifts are still required. Detailed specifications have been provided for each classification of ground support utilized in distinct types of ground, as summarized on Table 16-31.

Table 16-30: Summary of Rock Quality for Each Ground Support Type

Drift Type	Ground Condition Type	Rock Zone	Q Class	Q Range	Q Portion of Drifting	Support Category
Decline	1	Metavolcanic Weathered	Very Poor	0.1 to 1	19%	IV
	2		Poor	1 to 4	47%	III2
	3		Fair	4 to 10	24%	II2
	4		Good	>10	10%	I2
Main Ramp	2	Metavolcanic Fresh	Poor	1 to 4	19%	III2
	3		Fair	4 to 10	39%	II2
	4		Good	>10	42%	I2
Footwall Access	5	Metavolcanic Fresh	Poor	1 to 4	19%	III1
	6		Fair	4 to 10	39%	II1
	7		Good	>10	42%	I1
Stope Access	8	Metavolcanics/ Metasediments Fresh	Poor	1 to 4	19%	III1
	9		Fair	4 to 10	39%	II1
	10		Good	>10	42%	I1

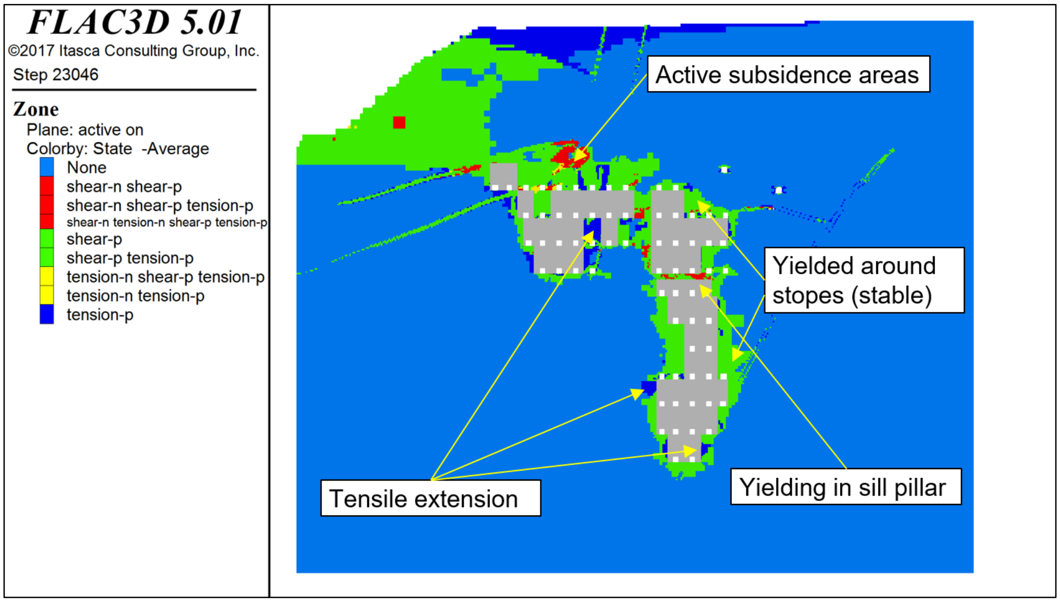
Source: SRK, 2017

Table 16-31: Summary of Ground Support Specifications for Each Category

Support Type	Support Categories	Bolt Length (m)	Bolt Spacing (m)	Other support	Ground Type
I	I1 - Spot Bolting	1.8	1.8	Split Sets	7,10
	I2 - Spot Bolting	2.4	1.5	Split Sets	4
II	II1 - Spot Bolting	1.8	1.8	Split Sets	6,9
	II2 - Spot Bolting	2.4	1.5	Grouted rebar w/ mesh	3
III	III1 - Sys Bolting	1.8	1.8	Split Sets w/mesh	5,8
	III2 - Sys Bolting	1.1	1.8	Split Sets	2
IV	IV - Sys Bolting	2.4	1.5	Fully grouted rebar, mesh, 9 cm shotcrete	1

Source: SRK, 2017

Remnant pillars and critical unmined areas have been checked for stability to prevent the collapse of ground which may cut off access to the ore or means of egress. The northwest-southeast cross section shown on Figure 16-18 provides an example of the numerical model results of predicted rock mass yielded areas. Ground conditions were assessed using the FLAC3D program to predict the area of excessive ground failure. No area of the design indicated excessive ground failure or massive stability issues.

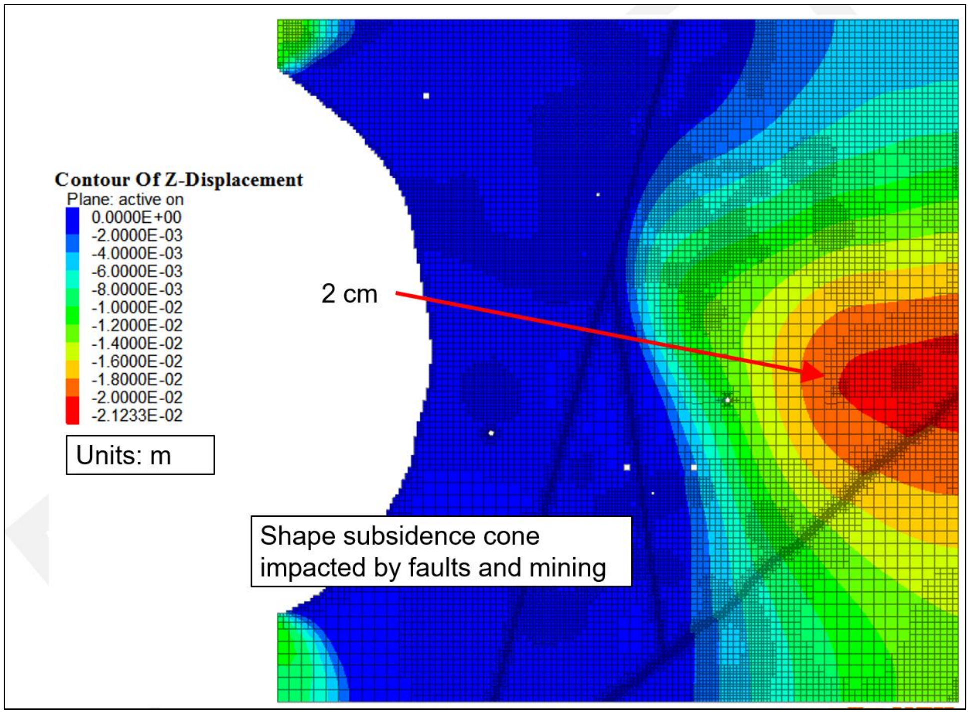


Source: SRK, 2017

Figure 16-18: Example of Plasticity Indicators along Northwest-Southeast Cross Section

The crown pillar above the mine has been checked for stability against ground collapse back to the surface. The estimated factor of safety is estimated using empirical crown pillar stability methods (Carter, 1980) to be significantly greater than 1.5, especially considering the top stopes will be tightly backfilled. The stability was confirmed by numerical modeling, which predicts no rock mass failure in the crown pillar area above mining and the stress arch above stopes is maintained.

Surface subsidence has been analyzed in the 3D numerical model to assess the potential for damage to the infrastructure above the mine or inflow of surface waters. Figure 16-19 shows the predicted subsidence above the underground mine. The extents of the nearby Snake pits are seen on the left-hand side of the figure as are the faults simulated in the model. The maximum predicted vertical subsidence above the mine will be less than 2 cm. The location of the maximum subsidence is influenced by movement along faults that intersect mining.



Source: SRK

Figure 16-19:Plan View of Vertical Displacements Representing Surface Subsidence

16.2.3 Hydrogeology and Mine Dewatering

The stratigraphy at the site is described in Section 7. The underground mining area extends to a depth of 390 m below land surface and it will be accessed by a decline from Snake Pit. This places the underground mine within the weathered and competent bedrock stratigraphic unit. An aspect of the conceptual hydrogeological model having most relevance to underground water management is that the weathered metavolcanics are more permeable than weathered metasediments, and unweathered basement rocks have a generally very low permeability – meaning that the weathered metavolcanic unit will likely be the predominant source of groundwater inflows to the underground mines (stopes and declines). Dewatering of the underground will be accomplished by capturing water entering the tunnels and pumping the water to the surface of the open pit where the water will be evacuated. Groundwater extraction rates from the underground have been estimated with the project groundwater model described in Section 16.1.8. Groundwater extraction rates from the underground mining are estimated to range from 1,000 to 3,000 m^3/d (12 to 35 L/sec). The majority of water requiring management is sourced from the decline and, specifically, at relatively shallow depths where the decline encounters weathered metavolcanic rocks. The timing and volume of extracted groundwater from the underground mining is expected to be manageable.

16.2.4 Geochemical

The objective of the underground geochemical program was to determine the metal leaching (ML) and ARD) potential of development rock that would be generated from the underground operations, including CRF from test programs that were done to evaluate the geotechnical and geochemical properties of materials proposed for underground backfill.

Previous Work

Development rock generated from mining at Haile has a range of ML/ARD potential, depending on lithology. For the underground work, the Schafer (2015) characterization categories are used.

The source of ARD is sulfide minerals, principally pyrite, deposited by the hydrothermal alteration that also deposited gold. Starting in 2008 and culminating in 2015, Schafer (2015) performed an extensive geochemical characterization program of existing and future mine development rocks to identify, manage and mitigate geochemical risks at Haile as part of the open pit plan. The characterization program included static testing of 4,911 samples as well as kinetic testing of nine samples of overburden and one tailings sample. The current development rock management plan employs three categories based on combinations of the total sulfur content (S_T) and net neutralization potential (NNP). The NNP is a measure of overall acid generation potential calculated as the difference between the neutralization potential (NP) and AP. The categories of potentially acid generating (PAG) development rock are:

- Red PAG – Strongly acid generating:
  - NNP < -31.25 kg $CaCO_3$/t;
- Yellow PAG – Moderately acid generating:
  - S_T > 0.2 % or NNP between 0 and -31.25 kg $CaCO_3$/t; and
- Green – Not acid generating:
  - S_T < 0.2 % or NNP > 0 kg $CaCO_3$/t.

The geochemical categories, and their associated management approaches, of development rock and saprolite within 15 m (50 ft) of the bedrock contact are summarized in Table 16-32.

Table 16-32: Current Open Pit, and adopted UG Overburden Classification at Haile

Operational Testing Criterion	Overburden Abundance	Characteristics	Proposed Management
Red PAG - strongly acid generating overburden			
Found in Metasediment Unit. For Metasediment, NNP < -31.25 kg/t as CaCO3	About 38% of Metasediment unit	When oxidized, contact water will have low pH (< 3.0) and very high metals, sulfate and acidity (>5,000 mg/L)	Stored in geomembrane encapsulated PAG cell, placed in lifts, compacted and Saprolite-lined outside perimeter to reduce oxygen supply
Yellow PAG - moderately acid generating overburden			
Found in Metasediment and Metavolcanic Bedrock Units and Saprolite. For bedrock, Total S > 0.2% or NNP <0 and NNP> -31.25 kg/t as CaCO3. For Saprolite within 50 feet of bedrock contact, Total S > 0.2% and NNP> -31.25 kg/t as CaCO3	About 22% of Metasediment unit, 6% of Metavolcanic unit, and 5% Saprolite	If allowed to oxidize, contact water will have low pH (3.0 to 4.0) and low to moderate metals (mostly Fe & Al)	Managed as above then may be placed in lifts as subaqueous pit backfill (2 lbs/t lime added as needed and 5-ft saprolite cover)
Green Overburden - not acid generating			
Found in Metasediment and Metavolcanic Bedrock Units, Saprolite and Coastal Plain Sands. For Bedrock, Total S < 0.2% and NNP > 0 kg/t as CaCO3. For Saprolite within 50 feet of bedrock contact, Total S < 0.2%. All Saprolite more than 50 feet above bedrock and all Coastal Plain Sand is Green Overburden.	40% Metasediment Unit, 94% Metavolcanics, 95% Saprolite and all Coastal Plain Sand	Contact water may have moderately acidic to alkaline pH (4.0 to 8.0), sulfate low (<1,000 mg/L) and metals non-detectable.	Placed in unlined overburden piles. Runoff will not require treatment assuming it meets storm water requirements as expected

Source: Schafer, 2015

Kinetic testing of “Red” and “Yellow” material confirmed the applicability of the development rock classification scheme based on results of static tests, which included:

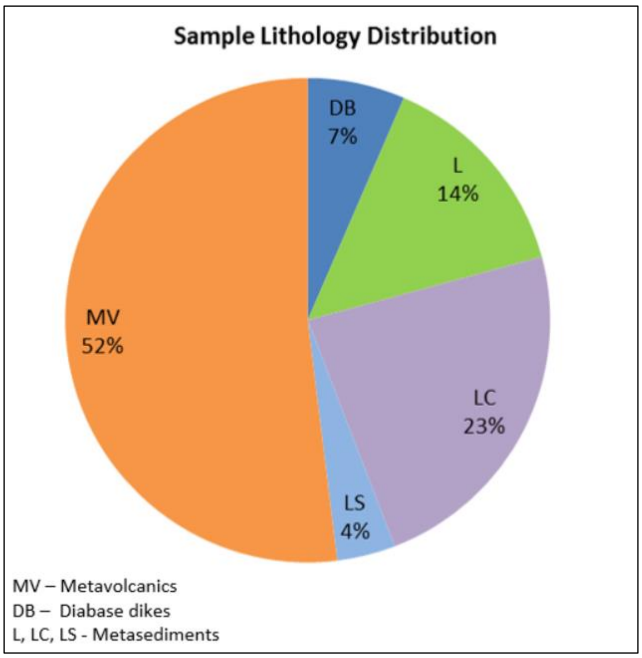
- Test work over 140 weeks on Red PAG samples developed low pH values (1.6 to 2.2) and released sulfate corresponding to one-third to one-half of the pyritic sulfur originally contained in the sample;
- Yellow PAG samples also had acidic pH values, though generally higher than Red PAG. High concentrations of metals were also released from these samples. Cumulative sulfate leached was much lower than for Red PAG samples owing to the lower levels of pyritic sulfur; and
- Green overburden samples maintained near neutral pH and also maintained low to non-detectable sulfate levels.

<u>Sample Collection - Development Rock</u>

The initial sampling program completed for the PEA level characterization involved collection of 24 samples, including eight that are representative of the Horseshoe deposit (SRK, 2016b). The FS-level study was a continuation of the site wide geochemical characterization by previous investigators and the PEA level characterization of the three underground targets (Horseshoe, Mustang and Mill Zone Deep) performed by SRK in 2016, and expanded the scope to provide detailed geochemical characterization data for development rock from the Horseshoe underground area. The FS SAP did not include collection and testing of tailings.

Based on the objectives of the FS-level characterization, 76 core samples were collected and selected for analyses. Samples were selected to provide the full range of the principal lithologies and to cover the vertical range of each area. All proposed samples are within approximately 50 m of the anticipated mining infrastructures.

The distribution of samples by lithology is shown in Figure 16-20. Samples were collected from all the lithologies within the potential mine infrastructure in approximately the same distribution as the anticipated volume of development rock estimated in the PEA (SRK, 2016a). Certain lithologies with potentially high ARD potential (such as pyritic laminated metasediment) were selected for analysis, even though they make up a small percentage of the total anticipated development rock stream. The design of the FS sampling and analysis program was documented in the FS SAP (SRK, 2016b).



Source: SRK

Figure 16-20: Horseshoe FS/PEA Lithological Distribution of Samples

Sample Collection - Backfill Testing

SRK's geotechnical engineers designed a backfill testing program to evaluate the properties of cemented rockfill (CRF) generated from a surface plant using aggregate (development rock) from the open pits and underground development rock (SRK, 2017). After the geotechnical testing was completed, 18 samples of the material were tested at SGS (Canada) Inc. in Vancouver B.C. for ML/ARD potential. Table 16-33 shows the batch numbers that were analyzed, the corresponding material type, and the volume of cement that was added during the geotechnical testing.

Table 16-33: Backfill Samples

Sample ID	Material Type	Cement Content (%)
1	NAG metasediment	3.86
2	NAG metasediment	1.94
3	NAG metasediment	5.78
4	NAG metavolcanic	3.79
5	NAG metasediment	1.90
6	NAG metasediment	5.66
7	PAG metasediment	3.85
8	PAG metasediment	1.93
9	PAG metasediment	5.76
10	NAG metavolcanic	3.84
12	NAG metavolcanic	5.74
14	NAG metavolcanic	1.96
15	NAG metavolcanic	5.83
16	NAG metasediment	3.75
17	NAG metasediment	1.88
19	NAG metasediment	4.14
20	Sand	2.07
21	NAG metavolcanic	6.60

Source: SRK

Summary of Analytical Results

Samples were analyzed for static geochemical characteristics using methods similar to those used for the overburden geochemical characterization program (Schafer, 2015). A range of methods was used for sulfur species and sources of NP to evaluate the various sources of AP and NP, and potential differences between analytical methods. Samples were analyzed for the following parameters:

- Paste pH (Sobek et al., 1978);
- Sulfate via hydrochloric acid (HCI) digestion with ICP-OES analysis;
- Total sulfur by LECO furnace;
- Total carbon by LECO furnace;
- Total inorganic carbon (HCl leach);
- Modified NP (Lawrence, 1989);
- NP (Sobek et al, 1978);
- Single addition net acid generation (NAG) test (AMIRA, 2002);
- Trace metal analysis, using aqua regia digestion with ICP-OES/MS analysis; and
- To obtain silicon data needed for the backfill geotechnical design, CRF samples were also analyzed for whole rock analysis by lithium metaborate fusion with x-ray fluorescence finish.

The underground scope of work did not include characterization of tailings. The underground program also did not include characterization of the chemical interaction between potentially reactive surface areas of the mine and groundwater that might produce contact water with low pH and/or elevated concentrations of metals, acidity and sulfate that may require treatment.

PEA Program

Results of the PEA analytical program indicated that only one of the five laminated metasediment samples from the Horseshoe deposit contained S_T above the Haile management criterion of 0.2% (at 0.34% S_T) for “Yellow PAG”. The PEA level samples diabase and metavolcanic samples from the

Horseshoe target were classified as not acid generating (NAG), based on both the Haile and industry classification criteria.

Paste pH

Paste pH of the development rock and backfill samples was alkaline, indicating relatively low concentrations of acid generating secondary minerals. The distribution of pH did not vary considerably by lithology. The average paste pH for the sampled development rock lithologies ranged from 9.05 to 9.6. The lowest paste pH measured was 7.17 for a laminated metasediment sample. The backfill samples exhibited significantly higher paste pH values ranging from 9.72 to 12.2, presumably due to the cement content which ranged from 1.9% to 6.6% of the sample by weight.

Sulfur

Sulfur species and concentrations were determined by a range of methods to provide a comprehensive evaluation of potential sources of acidity. Observations based on the sulfur data include:

- Total sulfur in development rock ranged from less than 0.005% to 2.45%. Average concentrations varied significantly by lithologies with values of 0.10%, 0.7%, 0.43% and 0.06% for the DB, L, LC/LS and MV, respectively;
- Almost all sulfur occurs as sulfide, especially at higher total sulfur concentrations (SRK, 2017). Schafer (2015) showed in the overburden characterization that sulfide S levels average 94% of total S values. The FS program work revealed that sulfide ranges from 96% to 100% of total sulfur values.
- The highest sulfide values measured in development rock (2.37% and 2.45%) were in laminated metasediment samples collected from planned tunnels in the ore body connecting two stopes;
- The majority of sulfate values in development rock were below the PQL of 0.01%. Only three samples (all metasediments) had measurable sulfate, with a maximum concentration of 0.04%. The highest sulfate values occurred in samples with elevated total sulfur concentrations (1.1% to 2.45%).
- The CRF rock samples exhibited significantly greater total sulfur and sulfide concentrations than FS development rock. Sulfide in CRF samples ranged from 0.01% to 4.42%, with an average value of 1.94%. It was determined that overburden materials collected for CRF backfill testing were more representative of red PAG rock than green non-PAG rock as originally intended.

Neutralization Potential

Neutralization potential was evaluated based on the concentration of total inorganic carbon as well as via the Sobek and Modified methods. Observations regarding the NP of the FS samples include:

- Total carbon is almost equal to total inorganic carbon (TIC), indicating that the concentration of organic carbon is low and that total carbon is a good approximation for TIC;
- Sobek NP values are generally slightly higher than Modified NP, which is normal as the Sobek method tends to dissolve a greater portion of the neutralizing alumino-silicate minerals.
- NP determined by the Modified method and calculated from TIC are very similar with only few exceptions;
- NP in the CRF samples is comparable to that of the FS development rock.

- Virtually all the available NP is derived from carbonate minerals;
- The exact composition of the carbonates was not determined, but calcite ($CaCO_3$), dolomite ($CaMg(CO_3)_2$) and siderite ($FeCO_3$) are all principal country rock mineral phases (Schafer, 2010a); and
- Sobek NP slightly overestimates NP since it tends to include a greater percentage of NP derived from silicate minerals, which vary widely in their contribution to NP.

Net Acid Generation

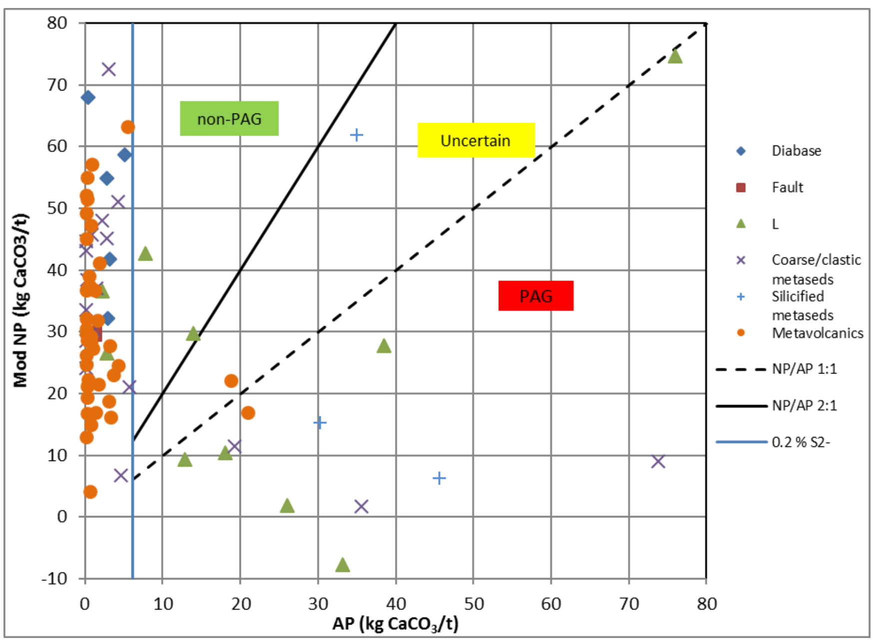
NAG pH values in development rock ranged from 2.45 to 6.92 and NAG pH values below 4.5 (indicative of PAG rock [AMIRA, 2002]) were observed in all lithologies except the DB (SRK, 2017). LS material had the highest fraction of PAG rock at 66%, followed by L (45.5%), LC (16.7%) and MV (5%).

NAG pH values in CRF material were comparable to development rock, ranging from 2.3 to 7.97, but a larger proportion of CRF samples (12 of 18) than development rock samples (12 of 78) reported NAG pH values less than 4.5. Overburden materials collected for CRF backfill testing were later found to be representative of red PAG rock rather than green non-PAG rock as originally intended.

ARD Potential

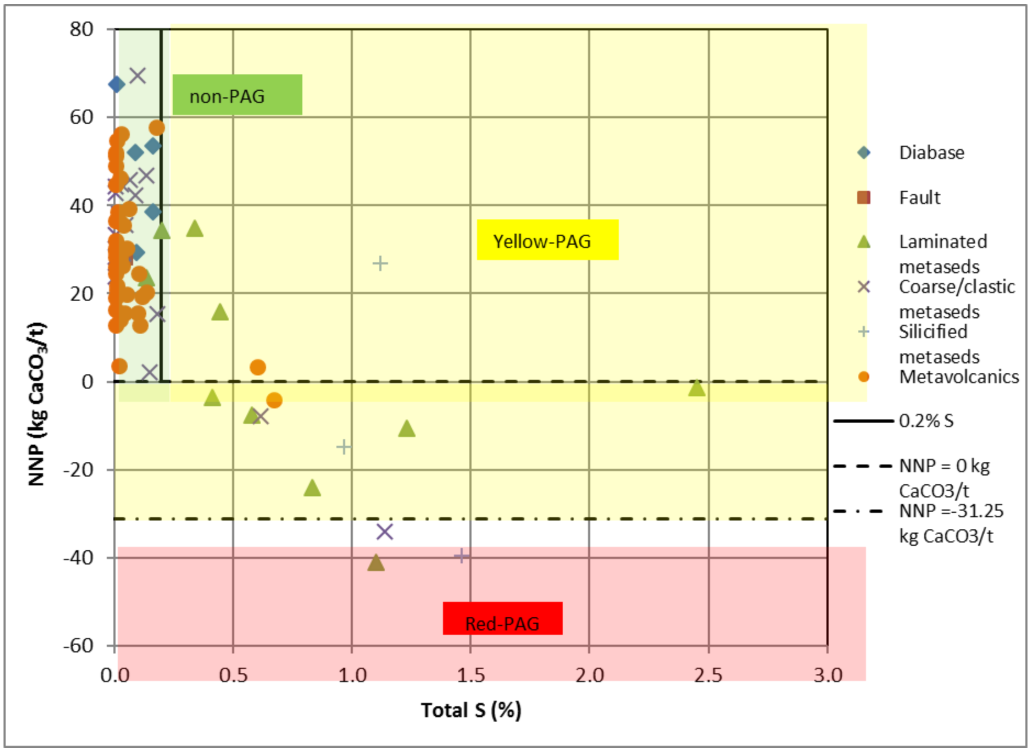
The AP of the samples was determined using the site-defined NNP criteria as well as literature criteria for NPR and NAG pH. The classification scheme of the samples is summarized in employing the color coding scheme (red, yellow, green) used by Haile for NNP and total sulfur. The ML/ARD potential of the samples was also categorized using industry criteria for acid generating material (Price, 2009) based on: NPR: ≤1 is PAG, between 1 and 2 is uncertain acid generation potential, and >2 is not PAG.

Figure 16-21 plots the Modified NP versus AP of development rock by lithology. Figure 16-22 presents a plot of NNP versus total sulfur using the Haile development rock criteria.



Source: SRK

Figure 16-21: NP/AP for Horseshoe Development Rock



Source: SRK

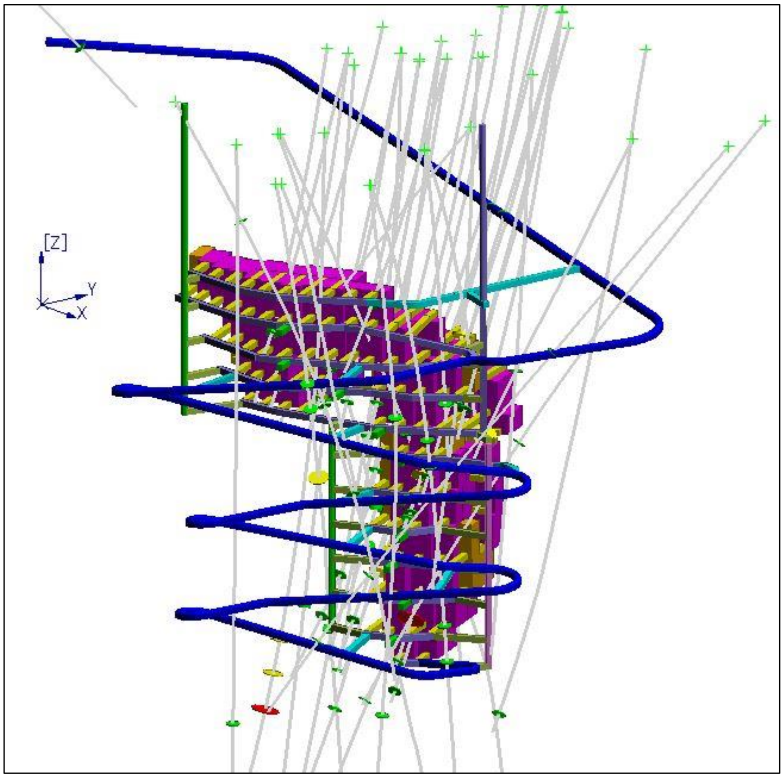
Figure 16-22: NNP vs. Total Sulfur for Horseshoe Development Rock

Observations related to the potential for acid generation using the Haile criteria for NNP and total sulfur include:

- There is no apparent correlation between NP/AP and depth of samples.
- The majority of MV rock (95%) is non-PAG.
- Slightly more than half (63.6%) of the L samples are PAG.
- Most of the coarse/clastic metasediments (LC) are non-PAG, with 2 of the 18 samples (11%) falling into the PAG categories.
- The metasediments with secondary silicification are predominantly PAG (red) or potentially PAG (yellow) (66%).
- Three samples plot on the boundaries of different PAG classification schemes, including:
    - DDH0368-645 to 653 (L) exceeds Haile S_T > 0.2% but is not PAG based on NPR and NAG pH.
    - DDH0542-855.0 to 857.7 (LS) and DDH0370-931.8 to 934.4 (MV) exceeds Haile S_T > 0.2%, but have NPR > 0 (Yellow PAG) and are not-PAG based on NAG results. NP/AP indicate the samples have “uncertain” acid generation potential.
- The CRF samples are predominantly Red PAG. The NAG metavolcanics of Batches 4, 10, 12, 14, and 15 are Green PAG, and Batches 3, 20 and 21 are Yellow PAG depending on the NNP method referenced.

The locations of the samples with ML/ARD potential were evaluated by plotting the geochemical characteristics, geology, and proposed infrastructure as shown in Figure 16-23, resulting in the following observations:

- The results of the geochemical analyses (i.e., PAG vs non-PAG) reflect the mineralization and geology. All of the PAG samples are located in the proposed cross cuts at the contact between the metavolcanics and metasediments.
- All samples in the drifts southeast of the ore body, decline and spiral ramp were non-PAG, with the exception of a single sample in the metavolcanic area of the infrastructure (DDH0370-931 to 934) which had uncertain AP (S_T of 0.6% and NPR of 3.3 kg $CaCO_3$/t). This sample was not PAG based on the NAG test.
- The spiral ramp in the preliminary FS infrastructure plan was changed to a series of long switch backs along the southeast face of the ore body. Only one sample near the new ramp configuration was classified as yellow PAG (DDH0370-931 to 934). However, given the borderline and ambiguous PAG classification of this sample and the average NNP and total sulfur for all MV samples of 28.9 kg CaCO3/t and 0.06%, respectively, the metavolcanic development rock from the Horseshoe infrastructure should be classified as non-PAG.
- The results of the FS geochemical characterization of the Horseshoe deposit are in good agreement with the results of the site wide overburden characterization.
- Overburden materials collected for CRF backfill testing were more representative of red PAG rock than green non-PAG rock as originally intended. Controls will need to be implemented to ensure that only green non-PAG rock is used for CRF backfill.



Source: SRK

Figure 16-23: Location of PEA and FS Geochemical Samples color coded using Haile criteria

In summary, potential development rock requiring selective handling and management is expected to occur entirely from the laminated sediments in close proximity to the proposed stopes.

Total Element Concentrations

Total element concentrations in Horseshoe deposit samples were analyzed by aqua regia digestion and then compared to crustal averages for similar rocks (Price, 1997). The objective of the comparison is to identify development rock that carries elevated metal concentrations and therefore might leach metals after development. A statistical analysis of the data is reported in SRK (2017). The ML process can occur under both neutral and acidic conditions, depending on the element, mineralogy, sulfide oxidation rate and other factors.

The metavolcanic rocks were compared to high calcium granitic rock (an intrusive analog to andesite). The metasediments were compared to sandstones and shales, as the lithology of the metasediments represents a range of depositional environments. Metasediment samples were enriched in Sb, As, Cu, Co, Au, Mn, Mo, Ni, P, S, Se, Ag, and Zn when compared to their average crustal abundances in sandstone. The elements with median concentrations and/or greatest number of samples that exceeded ten times the crustal abundance for sandstone included Co, Cu, Au, Mn, Mo. When compared to average crustal abundance of elements in siltstone, only Au and Ag were enriched in the metasediment samples. The silicified metasediment sediments (LS) displayed a unique distribution and concentration of metals, with all three samples exceeding ten times the average crustal concentration for Mo, Ni, Ag and Au. The metavolcanic samples were only enriched in Ag. Enrichment of As, Sb, Co, Hg and Se in metasediments with ARD potential are indicators of potential leaching of these metals from development rock that might have environmental impacts if the rock is not properly managed.

The characterization of metals in the overburden development rock (Schafer, 2010a) indicated that Au, S, As, Se, Mo were enriched in the laminated metasediments (excluding the silically altered lithology). The silically altered rocks were enriched in Au, S, Mo, As, Ag, Te and Se. The metavolcanic overburden rocks were enriched in Se, Au, S and As. The CRF materials show enrichment in As, Au, S, and Te.

Summary of Geochemical Testing Program

- Total sulfur in development rock ranged from less than 0.005% to 2.45%, equating to AP values from 0.16 to 76.6 kg $CaCO_3$/t. Average concentrations vary significantly by lithology with values of 0.10%, 0.7%, 0.43% and 0.06% for the DB, L, LC/LS and MV, respectively.
- Almost all sulfur occurs as sulfide, especially at higher total sulfur concentrations (SRK, 2017).
- The CRF rock samples exhibited significantly greater total sulfur and sulfide concentrations than FS development rock, with sulfide in CRF samples ranging from 0.01% to 4.42%, with an average value of 1.94%. Further investigation determined that overburden materials collected for CRF backfill testing were more representative of red PAG rock than green non-PAG rock.
- Virtually all the available NP is derived from carbonate minerals, so NP has been evaluated based on TIC rather than the Sobek (Sobek et al., 1978) or modified (Lawrence, 1989) methods.
- NP ranges from 0.83 to 70.8 kg $CaCO_3$/t.
- NP in the CRF samples is comparable to that of the FS development rock.
- Paste pH results for development rock and backfill samples were alkaline, indicating relatively low concentrations of acid generating secondary minerals.

The ML/ARD potential of the Horseshoe development rock and the CRF backfill material can be summarized as follows:

- The majority of MV rock (95%) is non-PAG.
- Slightly more than half (63.6%) of the L rock is PAG.
- The majority of the coarse/clastic metasediments (LC) are non-PAG.
- The metasediments with secondary silicification are predominantly PAG (red) or potentially PAG (yellow) (66%).
- All of the PAG samples are located in the proposed cross cuts at the contact between the metavolcanics and metasediments.
- Overburden materials collected for CRF backfill testing were more representative of red PAG rock than green non-PAG rock as originally intended. Controls will need to be implemented to ensure that only green non-PAG rock is used for CRF backfill.
- Development rock requiring selective handling and management is expected to occur entirely from the laminated sediments in close proximity to the proposed stopes.

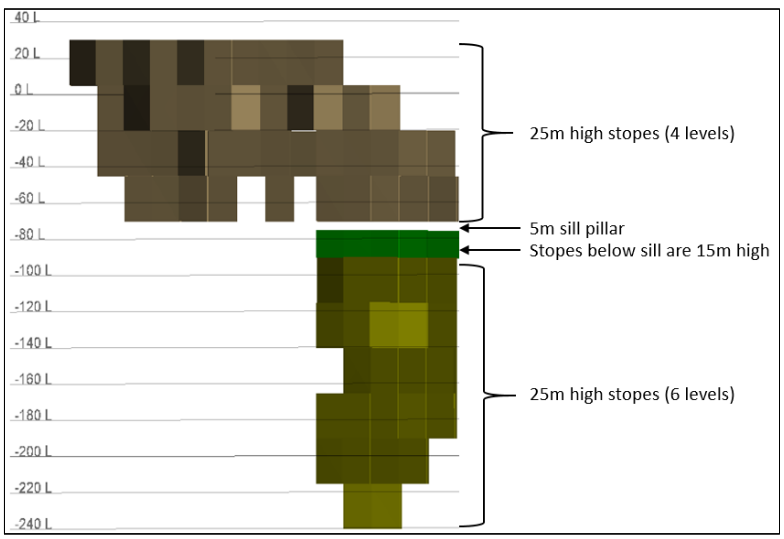
16.2.5 Stope Optimization

Stope optimization within Vulcan software was used to determine potentially economically minable material. Stope sizes used in the optimization were 15 m wide and a minimum of 10 m long and alternate stope heights were evaluated. Stope walls were vertical and wall dilution was not applied at the optimization stage. 20 and 25 m stope heights show approximately the same tonnage/grade profiles. At heights greater than 25 m the grade drops and the tonnage increases indicating the stopes include more dilution material and therefore the 25 m stope height was selected for the FS. This stope height analysis was undertaken separating the results in the upper Horseshoe area and the lower Horseshoe area, as there does appear to be different geologic and geotechnical controls in the two areas. Results show the lower area could be mined using slightly higher stopes, however for purposes of the FS, a constant 25 m stope height is used.

The economic benefit of various sill pillar locations was evaluated considering both material left in situ (based on stope optimization results) and time/cost to develop to the sill, as the mining method is a bottom up sequence. Results of the study indicated a single sill between the upper/lower Horseshoe zone boundary as the most viable.

Numerous stope optimization runs were completed to determine optimal level locations. In the upper mining block all stopes are 25 m in height. In the lower block the uppermost level below the sill is 15 m in height. Additionally, stope width was varied to 10 m to minimize planned dilution around the edges of the deposit.

Figure 16-24 shows the selected stope optimization configuration used for mine planning purposes.



Source: SRK

Figure 16-24: Stope Optimization Configuration Used for Mine Planning Purposes

Stope optimization results for the selected configuration show 3.15 Mt of mineralized material at a grade of 4.60 g/t Au (not including external dilution).

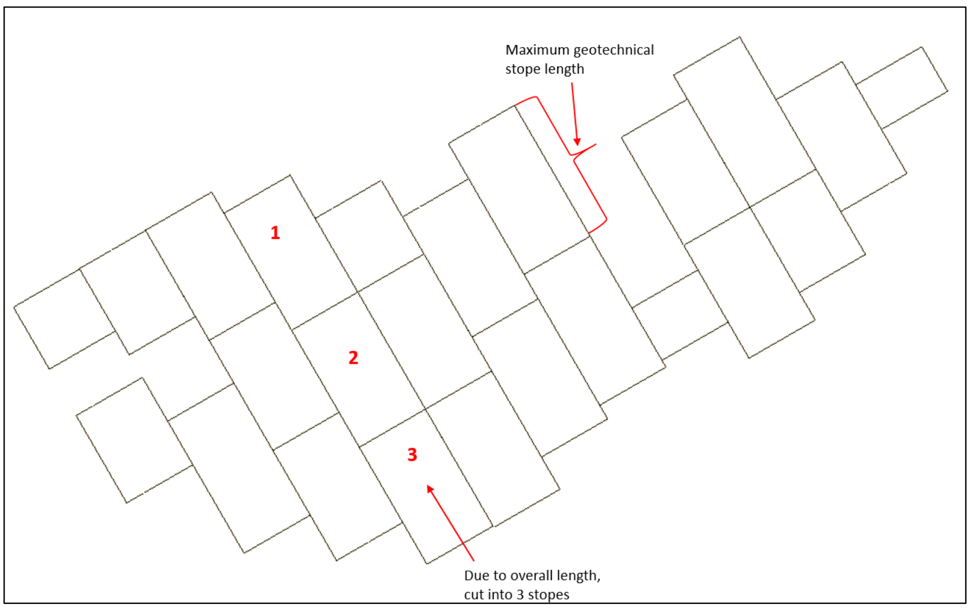
16.2.6 Mine Design

Stope optimization results were used as a basis for the underground mine design. The top of the Horseshoe mineralization is approximately 120 m below surface and extends to a depth of approximately 400 m below surface.

Stope Design

Stope optimizer shapes were used as a basis for the design work. Stope centerlines were generated and accesses into stopes were designed. Both top and bottom stope access are designed, as mucking will occur from the lower access and drilling/backfilling will occur from the upper access (with the exception of the stopes immediately below the -70L sill pillar). Stope accesses are expected to be in waste until they intercept the stoping block, but grade control will be used to determine the exact ore/waste boundary during mining.

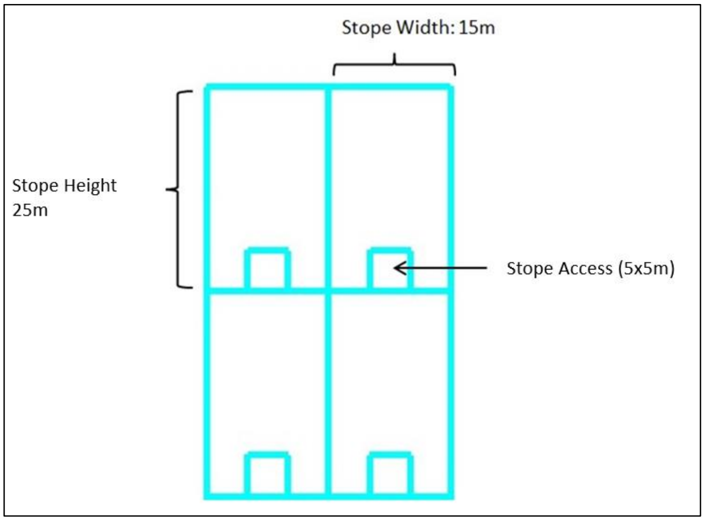
A typical level in the upper zone is made up of approximately ten stopes across, while the lower zone has approximately five stopes across. The length of stopes is limited by geotechnical stability and often several stope cuts are taken as shown in Figure 16-25. A primary/secondary stoping sequence will be used where, on any given level, primary stopes must be separated by a secondary stope. Extraction of the secondary stope can only occur after the two immediately adjacent primary stopes have been mined, backfilled and have had time to cure. Backfilling will be an integral part of the mining cycle as there is a limited quantity of stopes available on each level. Where possible stope cuts were aligned to minimize the requirement for CRF.



Source: SRK

Figure 16-25: Level Cross Section of Stopes (upper zone)

Figure 16-26 shows a cross section of the stopes.



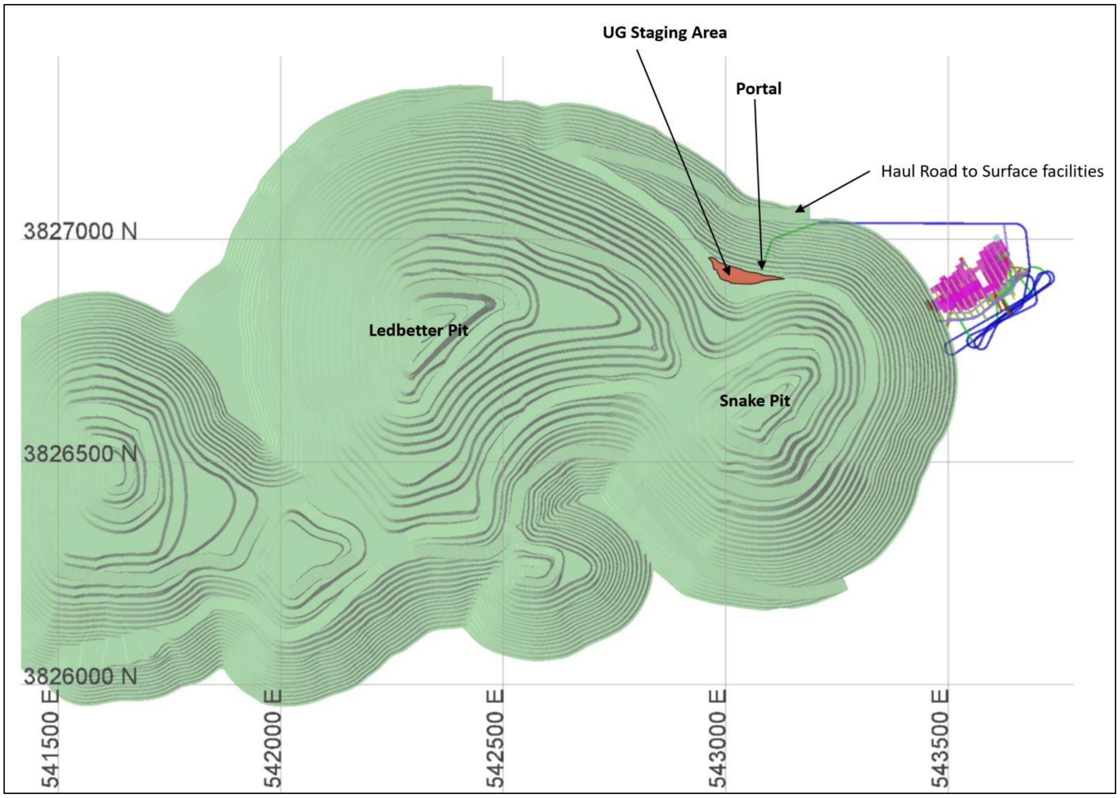
Source: SRK

Figure 16-26: Stope Cross Section

Stopes are developed using a slot as discussed in Section 16.2.9. Separate slot triangulations were not constructed for each stope, but the slot tonnage of each stope is separated out and a slot activity is used for scheduling.

Development Design

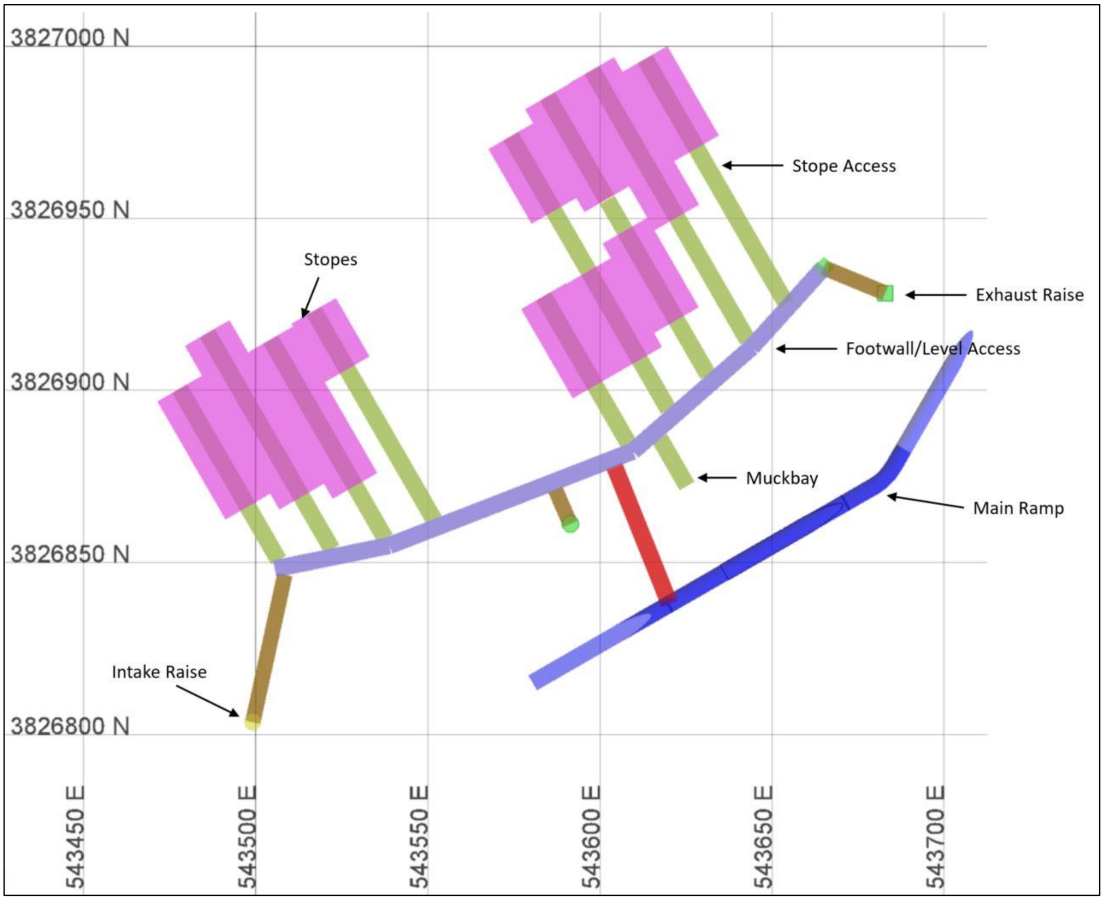
The underground mine is accessed via a decline from the surface. The decline portal is located on an open pit bench approximately 80 m below the natural surface as shown in Figure 16-27.



Source: SRK

Figure 16-27: Underground Portal Location

The stope accesses are connected to a level access located in the footwall in waste material. The level accesses are offset a minimum of 25 m from the stopes. The level accesses connect to the interlevel ramp system which is located in the footwall and is offset approximately 75 m from the stopes. On the southwest end of each level access there is a connection to a fresh air intake raise and on the northeast end the level access connects to an exhaust air raise. Figure 16-28 shows a typical level section.



Source: SRK

Figure 16-28: Typical Level Section

An additional development allowance was used for main ramps and level accesses to account for detail currently not in the design. These allowances are shown in Table 16-34. Where possible muck bays are re-used to minimize additional development.

Table 16-34: Additional Allowances Accounting for Detailed Mine Planning

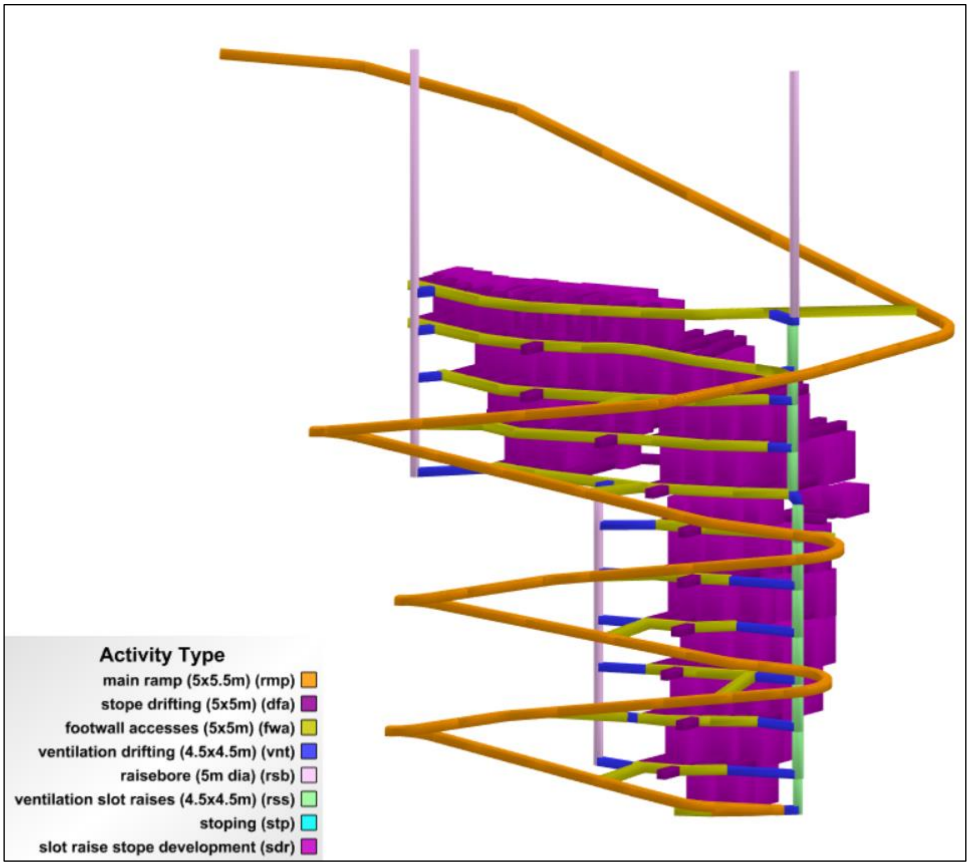
Main Ramps		
Muck bays - 4.5 m x 4.5 m by 16 m long, two per 500 m of ramp	648	m^3
Muck bays - high backs for loading – 7 m tall	268	m^3
Drill Bays - 4.5 m wide x 10 m long x 6 m high, one per 500 m of ramp	-	m^3
Electrical - 4.5 m x 4.5 m x 4 m long, one per 500 m ramp	81	m^3
Pump station - 4.5 m x 4.5 m x 16 m long, one per 500 m of ramp	324	m^3
Passing bays – 4 m wide, 35 m long, 4.5 m high, one per 500 m of ramp	685	m^3
Total additional development volume	2,006	m^3
500 m ramp volume	12,670	m^3
Additional Allowance for main ramps:	**16%**	
Footwall Access on Levels Assumption - levels are 150 m long (i.e., above sill)		
Pump station - none - drain back to main ramp	-	m^3
Electrical - 4.5 m x 4.5 m x 4 m long, one per level	81	m^3
Muck bays - 4.5 m x 4.5 m x 16 m long, one per level - designed in	-	m^3
Muck bays - high backs for loading – 7 m tall	122	m^3
Total additional development volume	203	m^3
Footwall Access one level volume	3,750	m^3
Additional Allowance for main ramps:	**5%**	**m^3**
Footwall Access on Levels Assumption - levels are 70 m long (i.e., below sill)		
Pump station - none - drain back to main ramp	-	m^3
Electrical - 4.5 m x 4.5 m x 4 m long, one per level	81	m^3
Muck bays - 4.5 m x 4.5 m x 16 m long, one per level - designed in	-	m^3
Muck bays - high backs for loading – 7 m tall	122	m^3
Total additional development volume	203	m^3
Footwall Access one level volume	1,750	m^3
Additional Allowance for main ramps:	**12%**	**m^3**

Source: SRK

All planned maintenance will be on the surface and underground shop facilities are not included in the design. The CRF facilities are also located on the surface and no additional infrastructure is required underground.

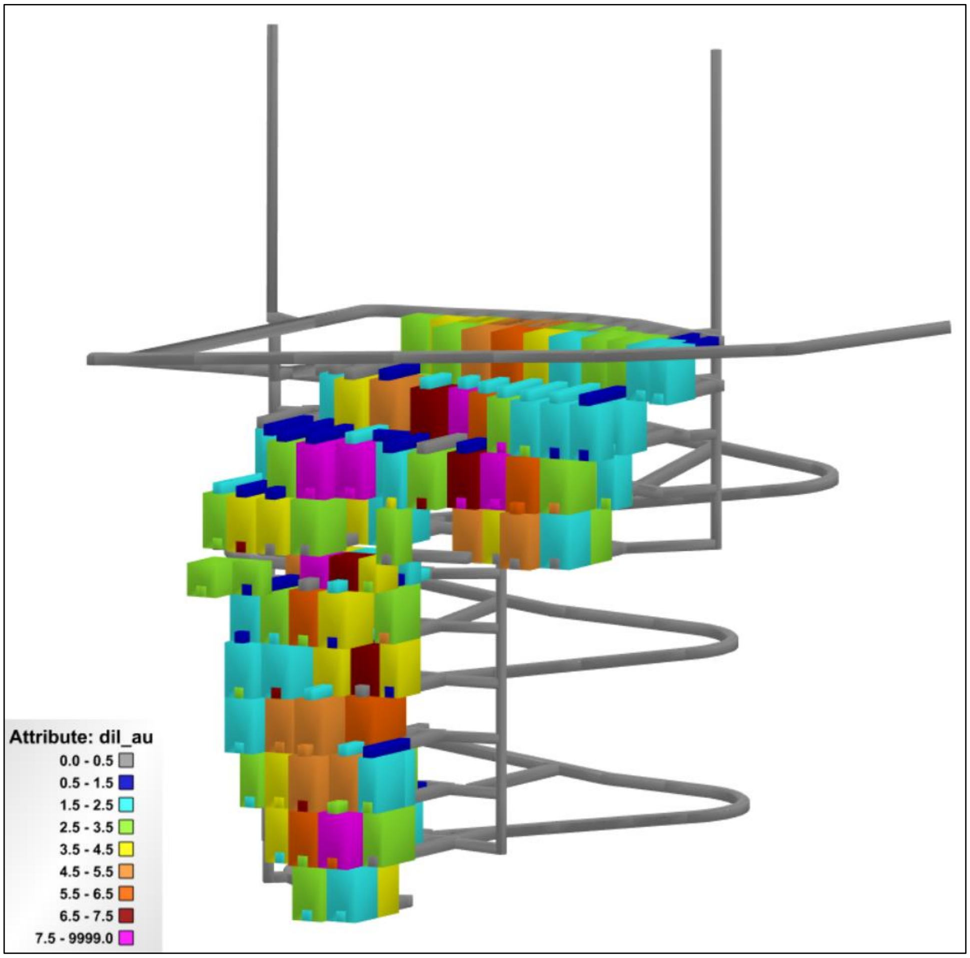
Where possible accesses/ramps have been designed to be located in the metavolcanics and away from known dikes. Where ramps must cross a fault/dike, the crossing is designed perpendicular to the structure to minimize the length of development through these structures.

Figure 16-29 and Figure 16-30 show the completed mine design colored by activity type and Au respectively.



Source: SRK

Figure 16-29: Horseshoe Completed Mine Design, colored by activity type (looking North)



Source: SRK

Figure 16-30: Horseshoe Completed Mine Design, colored by activity type (looking South)

Table 16-35 summarizes the mine design by activity type.

Table 16-35: Mine Design Summary – by Activity Type

General Summary	
Ore Tonnes (kt)	3,116
Ore Au (g/t)	4.38
Waste Tonnes (kt)	584
Total Tonnes Moved (kt)	3,700
Ore Summary	
Development Ore Tonnes (kt)	278.3
Stope Slot Development Tonnes (kt)	880.5
Stope Production Tonnes (kt)	1,957.2
Horizontal Development Summary	
Main Ramp Length – 5 x 5.5 (m)	2,845
Footwall Access Length – 5 x 5 (m)	2,058
Stope Access Drift Length in Ore – 5 x 5 (m)	3,958
Stope Access Drift Length in Waste – 5 x 5 (m)	2,467
Ventilation Drift Length - 4.5 x 4.5 (m)	562
Total Development Length (m)	**11,890**
Vertical Development Summary	
Raisebore Length – 5 m diameter (m)	478.7
Ventilation Slot Raise Length - 4.5 x 4.5 (m)	256.2

Source: SRK

Waste material within the design was characterized geochemically based on the block model described in Section 14.2.14. All rock characterized as red had sufficient grade to be considered ore. Only green and yellow waste are generated as summarized in Table 16-36.

Table 16-36: Mine Design Geochemistry Summary

Geochemical Breakout	
Green Waste Tonnes (kt)	477.2
Yellow Waste Tonnes (kt)	106.8
Total Waste Tonnes (kt)	**584.0**

Source: SRK

16.2.7 Productivities

Productivities were developed from first principles. Input from mining contractors, blasting suppliers and equipment vendors was considered for key parameters such as drilling penetration rates, blast hole size and spacing, explosives loading time, bolt and mesh installation time, etc. The rates developed from first principles were adjusted based on benchmarking and the experience and judgment of OceanaGold and SRK.

The productivity rates used for mine scheduling are shown in Table 16-37, followed by a description of the general and activity-specific parameters upon which the productivity rates are based.

Table 16-37: Productivity Rates

Activity Type	Dimensions	Rate (1)
Main Ramps – 1st three months of mining (single headings)	5 m x 5.5 m	4.55 m/d
Main Ramps – weathered zone (single headings)	5 m x 5.5 m	5.06 m/d
Main Ramps – below weathered zone (single headings)	5 m x 5.5 m	5.33 m/d
Level Development (single headings)	5 m x 5 m	6.09 m/d
Drifting top/bottom stope accesses (multiple headings)	5 m x 5 m	7.33 m/d
Slot Raise – Stope Development (2)		937 t/d
Stoping (2)		2,477 t/d
Raisebored Raise	5 m diameter	3.41 m/d
Slot Raise – Ventilation	4.5 m x 4.5 m	7.5 m/d
Backfilling (CRF)		1,000 m^3/d

Source: SRK
(1) All rates are per face. Multiple areas/faces are mined together to generate the production schedule.
(2) Includes drilling, blasting, and mucking.

General Parameters

General schedule parameters applicable to all underground mining activities are presented in Table 16-38.

Table 16-38: Schedule Parameters for Underground Mining

Schedule Parameters	Units	Value
Annual mining days	days/year	365
Mining days per week	days/week	7
Shifts per day	shifts/day	2
Scheduled shift length	hrs/shift	12
Scheduled deductions		
Travel to/from the underground working area from the surface	hrs/shift	1.00
Workplace examinations/equipment pre-shift inspections	hrs/shift	0.25
Lunch	hrs/shift	0.50
Breaks	hrs/shift	0.50
Total scheduled deductions	hrs/shift	2.25
Operating time (scheduled shift length less scheduled deductions)	hrs/shift	9.75
Effective time (operating time reduced to a 50-minute hour, i.e., multiplied by 83.3%)	hrs/shift	8.125

Source: SRK

Key assumptions regarding ore and waste material characteristics are detailed in Table 16-39.

Table 16-39: Material Characteristics

Characteristic	Units	Value
Ore in situ density	t/m^3	2.82
Ore swell	%	35
Ore loose density	t/m^3	2.09
Waste in situ density	t/m^3	2.77
Waste swell	%	35
Waste loose density	t/m^3	2.05

Source: SRK

For the purposes of developing productivity estimates, the ground support requirements detailed in Table 16-31 were simplified as described in Table 16-40.

Table 16-40: Simplified Ground Support Requirements

Drift Type	Ground Condition Type	Rock Zone	Class	% of Drifting	Operations	Bolt Length (m)	Bolt Spacing (m)	Support Type
Access Decline	1	Metavol-Weathered	Very poor	19%	V	2.4	1.5	Resin rebar, mesh, 9 cm shotcrete
	2		Poor	47%	IV	2.4	1.5	Resin rebar, mesh, 5 cm shotcrete
	3 & 4		Fair & Good	34%	III	2.4	1.5	Resin rebar w/ mesh
Main Ramp	2	Metavol-Fresh	Poor	19%	IV	2.4	1.5	Resin rebar, mesh, 5 cm shotcrete
	3 & 4		Fair & Good	81%	III	2.4	1.5	Resin rebar w/mesh
Footwall Access	5	Metavol-Fresh	Poor	19%	II	2.4	1.8	Split sets w/mesh
	6 & 7		Fair & Good	81%	I	2.4	1.8	Split sets
Stope Access	8	Metavol/Metased-Fresh	Poor	19%	II	2.4	1.8	Split sets w/mesh
	9 & 10		Fair & Good	81%	I	2.4	1.8	Split sets

Source: SRK

Main Ramp Development (long-term development openings)

The main ramp is 5 m wide by 5.5 m high with an arched back. It will be developed with a twin-boom jumbo drilling 41 mm diameter blast holes and 102 mm relief holes. All jumbo holes will be drilled 4.24 m in length, which allows for an effective advance rate of 3.81 m per round (90% pull). The drill pattern provides for 62 charged blast holes and five uncharged relief holes. Drilling times were calculated based on average penetration rates of 1.8 m/minute (min) for loaded holes and 1.0 m/min for reamed holes. A 10% re-drill factor was assumed.

Use of a bulk emulsion explosive was assumed at a powder factor of 1.16 kg/t. The blasting cycle time considered mobilization, charging and tying in of holes, clean-up, and demobilization.

Loading will be performed with a 6.2 m^3 (14 t) load-haul-dump unit (LHD) that will transport blasted rock to muck bays that are spaced 250 m apart. Load, maneuver and dump times were considered and a 95% bucket fill factor was assumed. Waste rock that is placed in a muck bay will be loaded into trucks and hauled to the surface (or to an empty secondary stope when available); however, the time associated with loading haul trucks is accounted for as a separate activity.

Ground support will be installed as specified in Table 16-40. Time allowances have been included for mobilization and setup, scaling, bolting/meshing/shotcreting as required, and demobilization. Cable bolts will be installed at intersections; however, the time associated with installing cable bolts is accounted for separately.

Utility installation includes piping lines, ventilation tube, electrical cable, messenger cable, and leaky feeder. Piping, ventilation and electrical utilities will be installed at the end of every other round.

As shown in Table 16-41, the standard main ramp single heading development rate is 5.33 meters per day (m/d).

Table 16-41: Main Ramp Development Rate

Task	Units	Value
Drilling	hrs/rd	3.64
Blasting	hrs/rd	1.40
Mucking	hrs/rd	1.43
Bolting and meshing	hrs/rd	4.85
Shotcrete	hrs/rd	0.22
Utilities	hrs/rd	2.42
Total cycle time	hrs/rd	13.96
Rounds per day	rds/d	1.40
Total advance rate	**m/d**	**5.33**

Source: SRK

During the first three months while development crews are ramping up, the main ramp development is de-rated to 4.55 m/d.

Initial main ramp development through the weathered zone uses more intensive ground support and is estimated at 5.06 m/d.

Level Access Drifts (medium-term development openings)

The level access drifts will be 5-m-wide by 5-m-high with flat backs. They will be developed with a twin-boom jumbo drilling 41-mm diameter blast holes and 102-mm relief holes. All jumbo holes will be drilled 4.24 m in length, which allows for an effective advance rate of 3.81 m per round (90% pull). The drill pattern provides for 63 charged blast holes and five uncharged relief holes. Drilling times were calculated based on average penetration rates of 1.8 m/min for loaded holes and 1.0 m/min for reamed holes. A 10% re-drill factor was assumed.

Use of a bulk emulsion explosive was assumed at a powder factor of 1.19 kg/t. The blasting cycle time includes mobilization, charging and tying in of holes, clean-up, and demobilization.

Loading will be performed with a 6.2 m^3 LHD that will transport blasted rock to muck bays that will be located, on average, 75 m from the advancing face. Load, maneuver and dump times were considered and a 95% bucket fill factor was assumed. Waste rock that is placed in a muck bay will be loaded into trucks and hauled to the surface (or to an empty secondary stope when available); however, the time associated with loading haul trucks is accounted for as a separate activity.

Ground support will be installed as specified in Table 16-40. Time allowances have been included for mobilization and setup, scaling, bolting/meshing/shotcreting as required, and demobilization. Cable bolts will be installed at intersections; however, the time associated with installing cable bolts is accounted for separately.

Utility installation includes piping lines, ventilation tube, electrical cable, messenger cable, and leaky feeder. Piping, ventilation and electrical utilities will be installed at the end of every other round.

As shown in Table 16-42, the standard level access single heading development rate is 6.09 m/d.

Table 16-42: Level Access Development Rate

Task	Units	Value
Drilling	hrs/rd	4.08
Blasting	hrs/rd	1.40
Mucking	hrs/rd	1.06
Bolting and meshing	hrs/rd	4.18
Utilities	hrs/rd	1.48
Total cycle time	hrs/rd	12.21
Rounds per day	rds/day	1.60
Total advance rate	**m/day**	**6.09**

Source: SRK

Through the Stope Development Drifts (short-term development openings)

The stope development drifts will also be 5 m wide by 5 m high with flat backs. Productivity parameters for drilling, blasting, mucking, ground support, and utilities are the same as for the level access drifts; however, a multiple heading advance rate of 7.33 m/d was used for the stope development drifts.

Slots– Stope Development

After top and bottom stope development drifts are established a slot will be developed at the far end of the stope. The slot consists of a conventionally blasted drop raise and 28 fan-drilled holes that will be slashed into the void that is created by the drop raise. Including the fan-drilled holes, the overall dimensions of the slot will be 15 m wide by 6 m long by 25 m high.

All blast hole drilling for the slot will be at a diameter of 114 mm (4.5 inches) using a down-the-hole (DTH) drill. A total of 50 holes will be required for the slot (22 holes for the drop raise and 28 holes for slashing). The estimated penetration rate for the DTH drill is 0.75 m per minute and the total drilling requirement is 1,088 m (including 10% re-drill).

The slot will be removed in a series of four blasts using a bulk emulsion product. The first two blasts will remove the bottom 14 m of the drop raise. The third blast will remove the remaining six meters at the top of the drop raise along with 14 of the fan-drilled slash holes on one side of the drop raise. The fourth and final blast will remove the remaining fan-drilled slash holes on the opposite side of the drop raise. The overall powder factor for the slot will be 1.39 kg/t based on a slot tonnage of 5,279 t. The blasting cycle time includes travel/set up time, charging and tying in of holes, clean up, and demobilization.

Slot ore will be mucked with a 6.2 m^3 LHD that will transport blasted ore to muck bays that will be located, on average, 100 m from the stope. Load, maneuver and dump times were considered and a 95% bucket fill factor was assumed. Ore that is placed in a muck bay will be loaded into trucks and hauled to the surface; however, the time associated with loading haul trucks is accounted for as a separate activity.

As shown in Table 16-43, the slot production rate is 937 t/d.

Table 16-43: Slot Production Rate

Task	Units	Value
Drilling	hrs/slot	30.40
Blasting	hrs/slot	19.20
Mucking	hrs/slot	28.24
Total cycle time	hrs/slot	77.85
Days per slot	days/slot	5.63
Total production rate	**t/d**	**937**

Source: SRK

<u>Stoping</u>

Stopes will be 25-m in height x 15-m in width and will have varying lengths (22.2-m on average). A tophammer longhole production drill will be used to fan drill the stope from the upper access drift. Blast holes will be 114 mm (4.5 inches) in diameter and the estimated drill penetration rate is 1.2 m/min. The total drilling requirement is 191 m per ring (including 10% re-drill) and the ore blasted per ring is 2,961 t.

Stope blasting will be performed in a series of three-ring blasts, the number of which will be a dictated by the length of the stope. Each three-ring blast will have a total of 39 charged holes (13 holes per ring). A bulk emulsion product will be used and the powder factor will be 0.60 kg/t. The estimated blasting cycle time includes travel/set up, charging and tying in of holes, clean up, and demobilization.

Stope ore will be mucked with a 6.2 m^3 LHD that will transport blasted ore to muck bays that will be located, on average, 100 m from the stope. Load, maneuver and dump times were considered and a 95% bucket fill factor was assumed. Ore that is placed in a muck bay will be loaded into trucks and hauled to the surface; however, the time associated with loading haul trucks is accounted for as a separate activity.

As shown in Table 16-44, the stope production rate is 2,477 t/d.

Table 16-44: Stope Production Rate

Task	Units	Value
Drilling	hrs	14.22
Blasting	hrs	6.68
Mucking	hrs	49.03
Total cycle time	hrs	69.94
Days	d	3.58
Total production rate	**t/day**	**2,477**

Source: SRK
Based on 9.0 m of advance, which is one three-ring blast totaling 8,883 tonnes.

<u>Main Ventilation Openings – Raisebored Raises</u>

Raiseboring and blind sinking will be used for the main ventilation openings. The one blind sink and two raisbored raises are contemplated by the mine plan will each have a diameter of 5.0 m and will have the following lengths.

- Exhaust raise from the 25L to the surface: 125 m
- Intake raise from the -70L to the surface: 216 m
- Intake raise between the -215L and the -70L: 138 m

Blind sinking rates are assumed to be 0.5 to 1.0 m/d. Overall, the raisebore rate used in the production schedule is 3.41 m/d.

<u>Ventilation Connections Between Levels – Drop Raises</u>

Conventional drop raising will be used to establish ventilation connections between level access drifts. The ventilation connections will be 4.5 m wide by 4.5 m long by 20.5 m high.

All blast hole drilling for the ventilation connections will be at a diameter of 114 mm (4.5 inches) using a DTH drill. A total of 22 holes will be required for the drop raise (16 charged blast holes and 6 uncharged relief holes). The estimated penetration rate for the DTH drill is 0.75 m per minute and the total drilling requirement is 631 m (including 10% re-drill).

The drop raise will be removed in a series of three blasts using a bulk emulsion product. The first two blasts will remove the bottom 13 m of the drop raise. The third and final blast will remove the remaining 7.5 m at the top of the drop raise. The overall powder factor will be 3.57 kg/t based on a tonnage of 1,150 t. The blasting cycle time includes travel/set up time, charging and tying in of holes, clean up, and demobilization.

Waste from the drop raises will be mucked with a 6.2 m^3 LHD that will transport the blasted rock to muck bays that will be located, on average, 100 m from the bottom of each raise. Load, maneuver and dump times were considered and a 95% bucket fill factor was assumed. Waste that is placed in a muck bay will be loaded into trucks and hauled to the surface (or to an empty secondary stope when available); however, the time associated with loading haul trucks is accounted for as a separate activity.

As shown in Table 16-45, the ventilation drop raise advance rate is 7.5 m/d.

Table 16-45: Ventilation Drop Raise Advance Rate

Task	Units	Value
Drilling	hrs/raise	16.19
Blasting	hrs/raise	13.09
Mucking	hrs/raise	6.46
Total cycle time	hrs/raise	35.74
Days per raise	days/raise	2.73
Total advance rate	**m/day**	**7.50**

Source: SRK

16.2.8 Mine Production Schedule

The Horseshoe underground mine production schedule is based on the productivity assumptions shown in Table 16-24. The schedule was completed using iGantt scheduling software and is based on mining operations occurring 365 days/year, 7 days/week, with two 12-hr shifts each day. A production rate of 1,924 t/d was targeted with ramp-up to full production as quickly as possible.

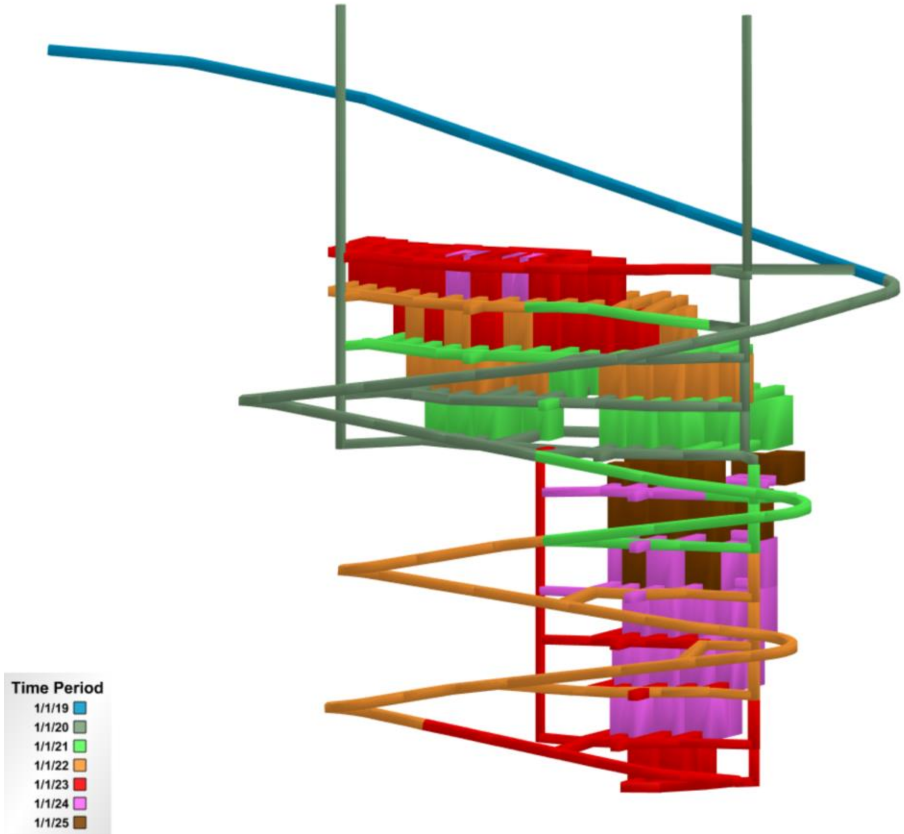
The iGantt scheduling work includes placing CRF and uncemented backfill in the mined-out stopes. Allowances were included for the time that will be required to cure the cement binder in the CRF. Specifically, a delay of seven days was assumed prior to operating equipment on CRF and a 14-day delay was assumed prior to mining adjacent to a CRF filled stope.

Table 16-46 and Figure 16-31 presents the annual mining schedule based on the aforementioned scheduling assumptions.

Table 16-46: Horseshoe Mine Production Annual Mining Schedule

Year	Mineralized Tonnes (kt)	Au (g/t)	Waste Tonnes (kt)	Backfill Volume (m^3)
2018	-	-	-	-
2019	-	-	47.9	-
2020	44.6	4.54	181.4	3,647
2021	701.1	4.42	95.6	223,912
2022	701.1	4.77	116.7	252,960
2023	701.2	4.03	113.8	249,620
2024	703.2	4.46	28.6	248,038
2025	265.0	3.95	0.0	106,909
Total	**3,116.0**	**4.38**	**584.0**	**1,085,086**

Source: SRK



Source: SRK

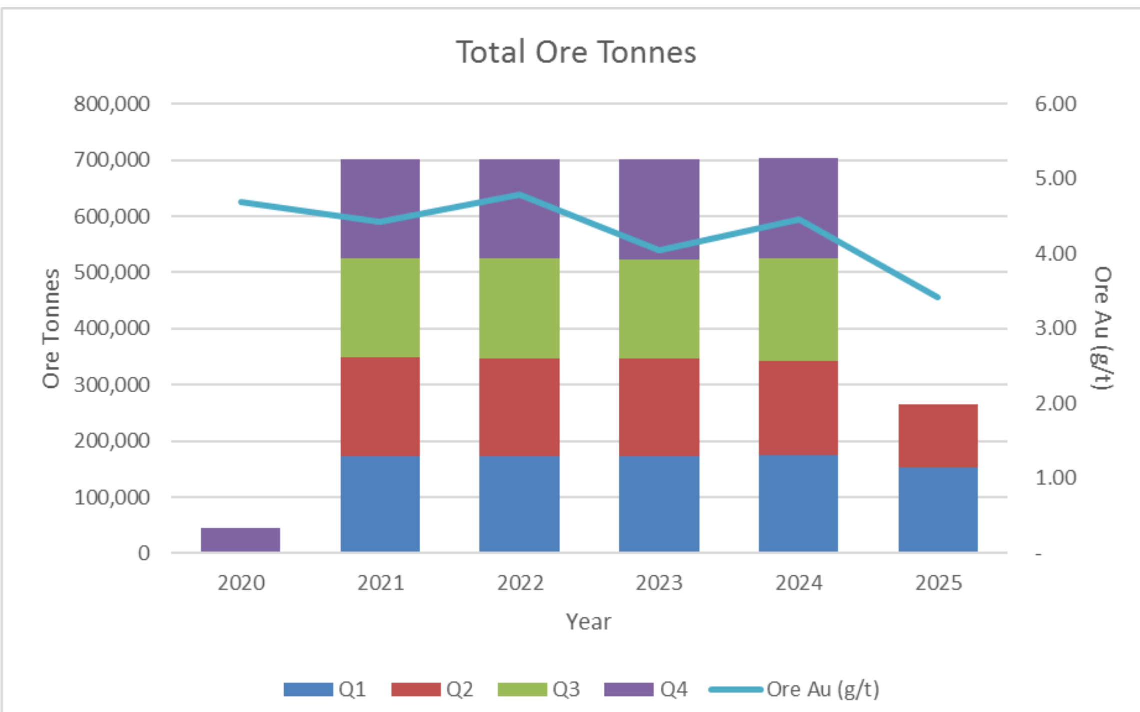
Figure 16-31: Mine Production Schedule Colored by Year

The underground permitting process is expected to be complete by mid-2019 and, accordingly, underground portal development is scheduled to begin on July 1, 2019. Prior to portal development, open pit mining in Snake Pit will have progressed sufficiently to allow access to the portal location. Portal development is assumed to take six weeks after which mining of the decline access can begin. First production from the stopes is scheduled to occur in December, 2020 and will last through mid-year 2025 based on the current Mineral Reserves.

Figure 16-32 through Figure 16-35 present additional summary information for the mine production schedule. Table 16-52 shows additional items in the production schedule summarized using the same timeframe as the economic model (quarterly, yearly).

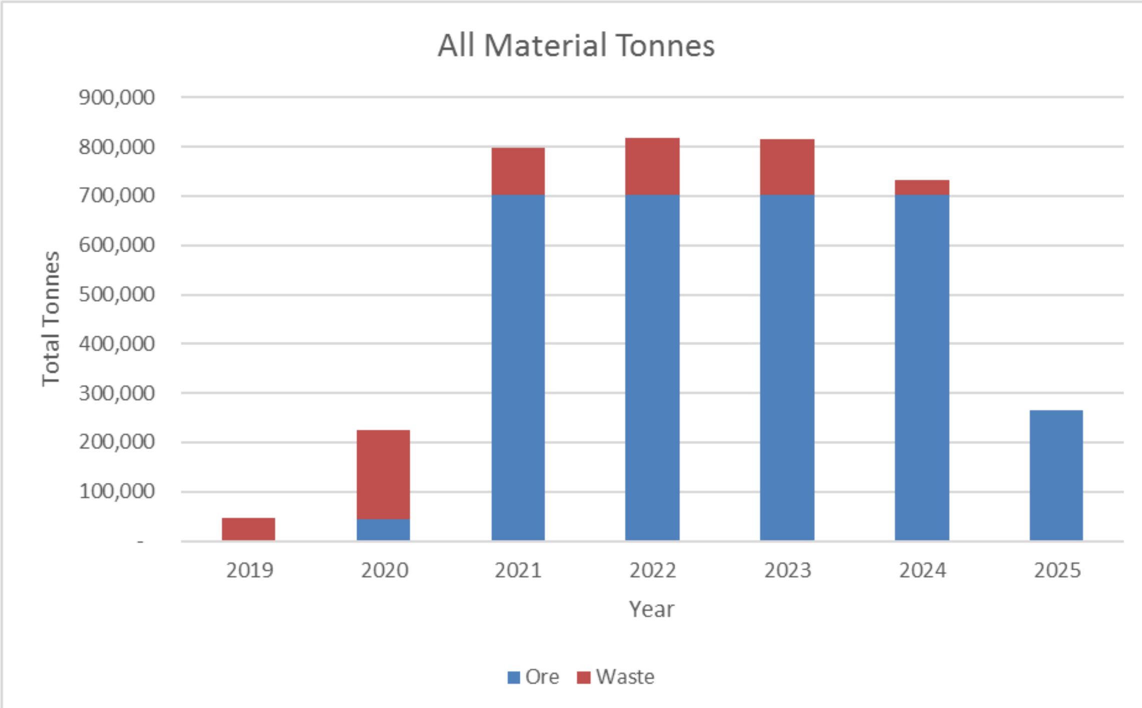
Source: SRK

Figure 16-32: Ore Tonnes by Activity Type



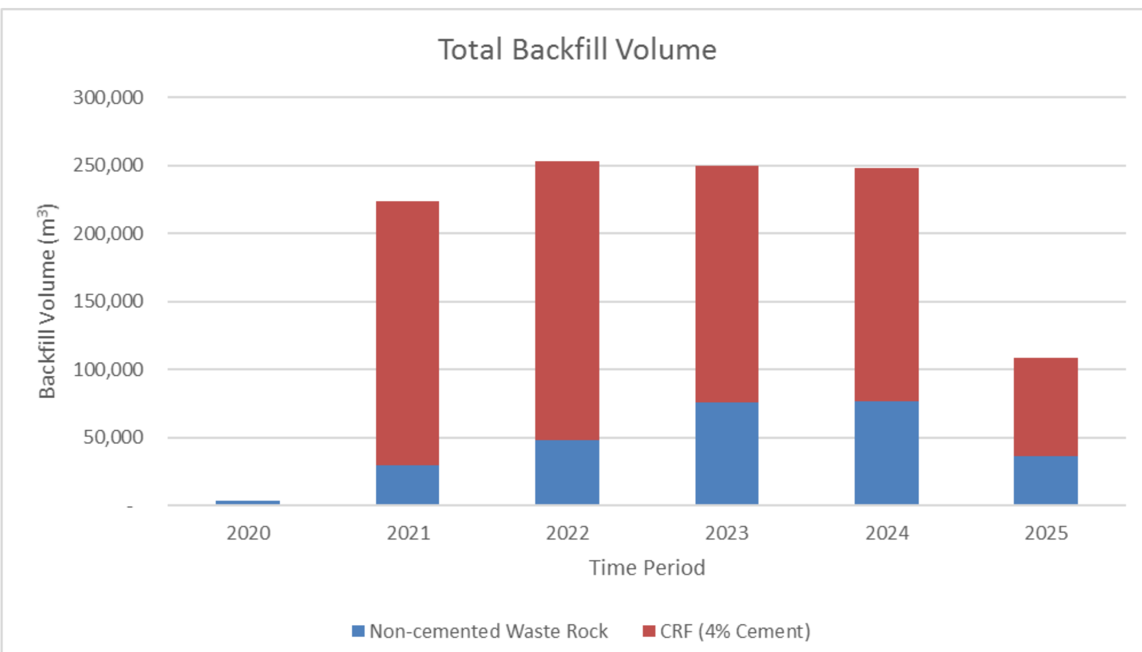
Source: SRK

Figure 16-33: Ore Tonnes and Grade



Source: SRK

Figure 16-34: All Material Tonnes



Source: SRK

Figure 16-35: Backfill Volume Type

Table 16-47: Detailed Mine Production Schedule

		2019		2020				2021	2022	2023	2024	2025
Economic Model Period		1	2	3	4	5	6	7	8	9	10	11
Time Period	Totals	8/1/2019	10/1/2019	1/1/2020	4/1/2020	7/1/2020	10/1/2020	1/1/2021	1/1/2022	1/1/2023	1/1/2024	1/1/2025
General Summary:												
Ore Tonnes/day (t/d)		-	-	-	-	-	484	1,921	1,921	1,921	1,920	1,264
Ore Tonnes (t)	3,116,039	-	-	-	-	-	44,571	701,063	701,057	701,200	703,185	264,960
Ore Au (g/t)	4.38	-	-	-	-	-	4.69	4.42	4.79	4.05	4.47	3.42
Waste Tonnes (t)	584,022	16,818	31,034	40,251	35,539	48,248	57,353	95,613	116,748	113,821	28,598	-
Total Tonnes Moved (t)	3,700,061	16,818	31,034	40,251	35,539	48,248	101,923	796,677	817,807	815,023	731,784	264,960
Ore Breakout:												
Development Ore Tonnes (t)	278,345	-	-	-	-	-	35,349	91,611	44,632	51,030	55,723	-
Stope Slot Development Tonnes (t)	880,534	-	-	-	-	-	6,409	179,599	192,500	192,081	198,296	111,649
Stope Production Tonnes (t)	1,957,160	-	-	-	-	-	2,813	429,854	463,927	458,089	449,163	153,310
Development Breakout:												
Main Ramp Length – 5 x 5.5 (m)	2,845.0	237	445	300	374	45.2	-	350	790	304	-	-
Footwall Access Length – 5 x 5 (m)	2,058.3	-	-	129	115	434	81.2	409	403	427	60.2	-
Stope Access Drift Length in Ore – 5 x 5 (m)	3,958.0	-	-	-	-	-	502	1,302	635	727	792	-
Stope Access Drift Length in Waste – 5 x 5 (m)	2,466.7	-	-	-	-	-	665	536	326	658	283	-
Ventilation Drift Length - 4.5 x 4.5 (m)	562.4	-	-	41.8	13.0	80.7	23.8	59.3	129	130	84.7	-
Total Development Length (m)	20,295	237	445	471	502	560	1,272	2,656	2,283	2,246	1,220	-
Vertical Development Breakout:												
Raisebore Length – 5 m diameter (m)	478.7	-	-	124	-	173	42.8	-	-	138	-	-
Ventilation Slot Raise Length - 4.5 x 4.5 (m)	256.2	-	-	26.4	16.0	25.0	25.0	45.0	75.0	43.8	-	-
Backfill Breakout:												
Total Backfill Volume (m^3)	1,085,086	-	-	-	-	-	3,647	223,913	252,959	249,620	248,038	108,612
Non-Cemented Waste Rock (m^3)	270,406	-	-	-	-	-	3,647	29,869	48,221	75,548	77,068	36,053
CRF - 4% Cement (m^3)	814,681	-	-	-	-	-	-	194,044	204,738	174,072	170,970	72,560
Haulage Breakout:												
Average Ore Haul Distance - One Way Distance (m)		-	-	-	-	-	2,462	2,518	2,343	2,320	3,272	2,458
Average Ore Cycle Time - Roundtrip (min)		-	-	-	-	-	31.6	32.4	29.8	29.6	43.2	31.7
Average Ore Haul Speed (km/hr)		-	-	-	-	-	9.4	9.3	9.4	9.4	9.1	8.0
Average Waste Haul Distance - One Way Distance (m)		1,346	1,574	1,849	2,220	2,633	2,700	1,238	978	1,133	600	600
Average Waste Cycle Time - Roundtrip (min)		15.2	18.0	21.2	26.3	31.9	32.7	15.4	13.8	16.1	8.4	8.4
Average Waste Haul Speed (km/hr)		10.6	10.5	10.5	10.1	9.9	9.9	9.4	8.7	8.6	8.6	8.6
Geochemical Breakout (1):												
Green Waste Tonnes (t)	477,214	16,818	31,034	40,221	32,969	43,791	41,855	79,258	95,240	81,316	14,712	-
Yellow Waste Tonnes (t)	106,807	-	-	30	2,569	4,457	15,498	16,355	21,507	32,505	13,887	-
Red Waste Tonnes (t)	-	-	-	-	-	-	-	-	-	-	-	-

Source: SRK
(1) Geochemical classification types discussed in section 16.2.4.

16.2.9 Mining Operations

Mine Access

A mine access and material handling tradeoff study was completed for the Horseshoe deposit. Options considered included shaft access with ore hoisting, decline access with conveyor haulage, and decline access with truck haulage. The tradeoff study demonstrated that, given the deposit depth, production rate, and anticipated mine life, the economically superior option is decline access with truck haulage.

The upper portion of the 5.0 m wide by 5.5 m high access decline is expected to be in weathered rock and therefore will require an increased level of ground support. After the decline has passed through the weathered rock, a less intensive level of ground support will be required. The decline is designed at a maximum gradient of 14%. A turning radius of 25 m was used, which is suitable for the underground haul trucks contemplated for the operation.

The portal for the access decline will be located on an open pit bench approximately 80 m below the natural surface, thereby eliminating the need to develop the access decline through saprolite. The portal construction will consist of scaling and bolting/screening and application of shotcrete as necessary to support and create a safe surface above the mine portal. A structurally sound corrugated pipe style liner with supports as necessary will be constructed for the first 200 feet of portal or as dictated by the rock conditions. Ventilation, power, water discharge, supply water, and communications will be installed at the portal and carried down the decline to support the development operation. An all-weather gravel surface will be established at the portal and portal bench area and drainage will be maintained away from the portal entrance to minimize water entering the portal and decline from the bench area.

Secondary egress will be via 5.0 m diameter raisebored ventilation raises equipped with emergency hoisting.

Stoping

Stopes will be mined using the sublevel open stoping method. Individual stope blocks are designed to be 15 m wide, up to 30m long, and will have a transverse orientation. Levels are spaced 25 m apart and each stope block will have a top and bottom access (5 m x 5 m flat back drifts).

Stopes will be drilled downward from the top access using 114 mm diameter holes (stope slots will be drilled with a DTH drill and stope production rings will be drilled with a tophammer drill). A bottom up, primary/secondary extraction sequence will be followed. Primary stopes will be backfilled with CRF and secondary stopes will be backfilled with RoM waste from the underground and open pit operations.

Stope extraction will occur in two steps. During the first step, a slot will be mined at the far end the stope using a drop raise and 28 fan-drilled slash holes. The slot is required to create sufficient void space for the remainder of the stope to be blasted. During the second step, production rings will be blasted three rows at a time (13 blast holes per ring) until the stope is completely extracted. The number of three-row blasts in a given stope will depend on the length of the stope. All blasting will be performed with bulk emulsion.

Ore will be remotely mucked from the bottom stope access using a 14-t LHD (6.2m^3). Cable bolts will be installed at the stope brow to ensure stability. The LHD will transport the ore to a muck bay to maximize the efficiency of the stope mucking operations. A second 14-t LHD and a fleet of 40-t haul trucks will be used to transport ore from the muck bays to the surface. Multiple muck bays will be used on each level to avoid interference between the stope loader and the haul trucks.

At the surface, the haul trucks will dump onto a RoM ore stockpile and will then travel to an adjacently located backfill plant to be loaded with CRF. After being loaded, the haul trucks will return to the underground mine and will dump the CRF into a muck bay near the top of an empty primary stope. After dumping the load of CRF at the muck bay, the haul truck will return to the producing level to once again be loaded with ore. A 7-t LHD will be used to transport the CRF from the muck bay to a dumping point at the top access of the empty stope.

Lateral Development

Lateral development includes interlevel ramps, level accesses, stope accesses, and short connecting drifts for ventilation. The interlevel ramp system will be a continuation of the main access decline and will have the same dimension (5.0 m wide by 5.5 m high with an arched back) and the same maximum gradient (14%). Level accesses and stope accesses will be 5 m wide by 5 m high with a flat back and will be mined higher at the muck bays to allow the haul trucks to be loaded by the LHD. The short connecting drifts for ventilation will be 4.5 m wide by 4.5 high with a flat back.

Interlevel ramps and levels accesses will be located in the footwall and have been designed to avoid crossing fault zones to the maximum extent possible. Stope accesses are oriented perpendicular to the strike of the orebody.

The lateral development is sized for the operation of the mining equipment fleet that has been selected for the operation. The stope accesses are wide enough to allow 40-t haul trucks to reverse into the stopes for direct tipping of backfill, provided that the back is mined higher (> 6.5 m) at the tipping point to allow the truck bed to be raised to the full height. The development profiles include allowances for ventilation ducting and services.

Vertical Development

Raiseboring and blind sinking will be used for the main ventilation openings. The one blind sunk and two raisbored raises contemplated by the mine plan will each have a diameter of 5.0 m and will have the following lengths.

- Exhaust raise from the 25L to the surface: 125 m
- Intake raise from the -70L to the surface: 216 m (equipped with an emergency hoist)
- Intake raise between the -215L and the -70L: 138 m (equipped with an emergency hoist)

Following discussions between OceanaGold and a number of underground mining contractors raiseboring of all the shafts may not be possible particularly in the upper weather sapolite portion of the geologic units. It is proposed that the exhaust raise be blind sunk from surface to the 25 L. The intake raise to the -70 L, would be blind sunk from surface to competent rock and the remainder raisebored.

Conventional drop raising will be used to establish ventilation connections between level access drifts.

The ventilation drop raises will be 4.5 m wide by 4.5 m long by 20.5 m high.

Truck Haulage

The mine plan assumes that 14-t LHDs will load 40-t haul trucks from muck bays that will be strategically located throughout the development workings. Ore and waste haulage distances and cycle times were calculated using the haulage profile module in Vulcan and are based on estimated underground truck speeds as shown in Table 16-48. The outputs from the Vulcan haulage profile module are a one-way haulage distance and an average truck cycle time (round trip).

Table 16-48: Truck Hauling Speeds

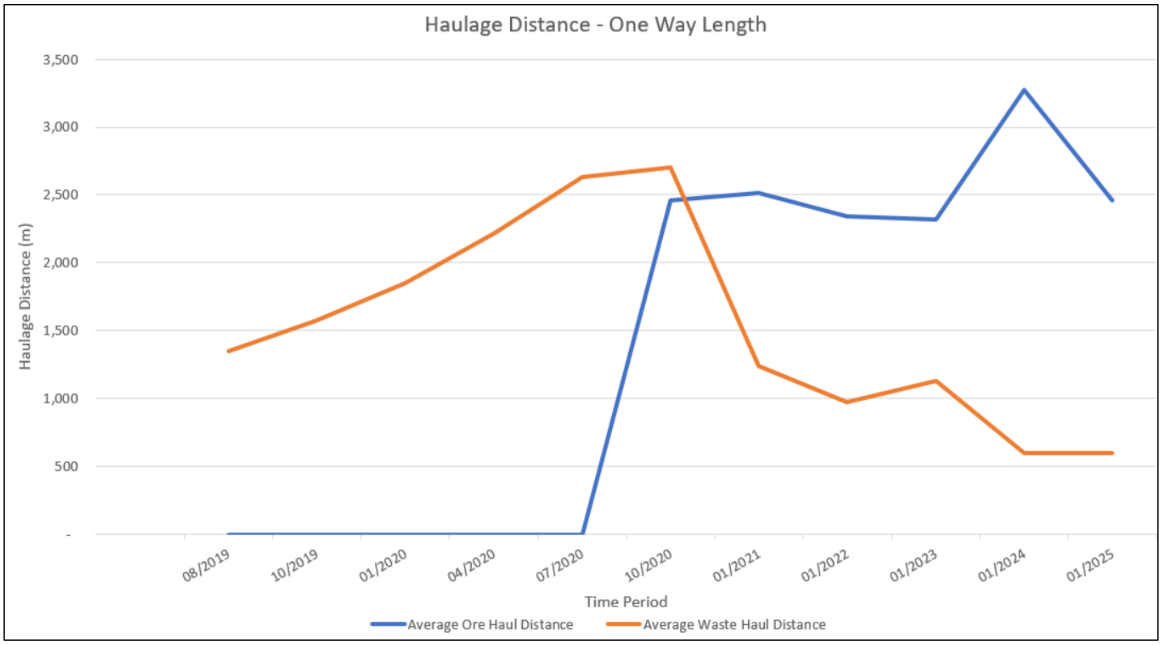
	Road Grade (%)	Speed [1] (km/hr)
Loaded	0-2.5	11.0
	2.5-5.0	10.5
	5.0-7.5	10.3
	7.5-10.0	10.2
	10.0-12.5	10.1
	12.5-15.0	7.4
	15.0-20.0	7.4
Empty	0-2.5	11.0
	2.5-5.0	10.5
	5.0-7.5	10.5
	7.5-10.0	10.5
	10.0-12.5	10.5
	12.5-15.0	9.0
	15.0-20.0	7.5

Source: SRK
(1) Uphill and downhill assumed speeds the same.

The ore haulage distances were evaluated from the mine production schedule. Based on this evaluation, ore haulage pathways were created to approximate the location of ore development and stope mining in each time period. Vulcan haulage profiles were then used to generate a one-way ore haulage distance and an average cycle time (round trip) using the speed parameters shown in Table 16-48.

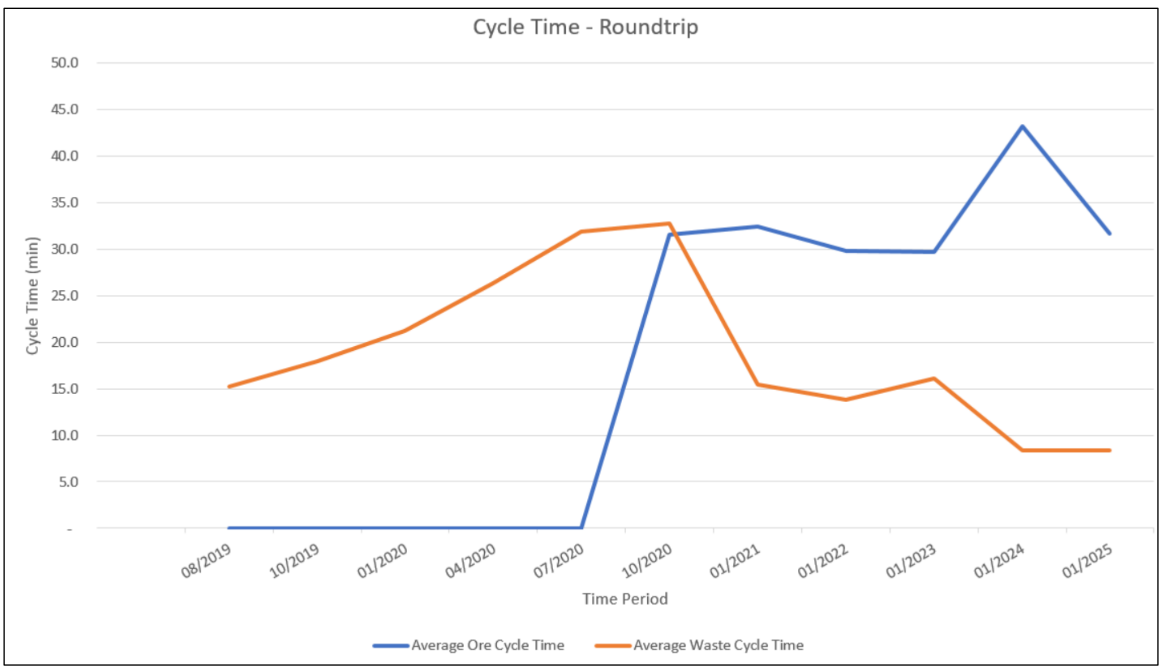
The availability of mined-out secondary stopes was evaluated to determine the quantity of non-cemented waste rock that will be placed underground during each time period. Based on this evaluation, waste haulage pathways were created to approximate the location of development waste mining and waste rock dumping for each time period. Vulcan haulage profiles were then used to generate a one-way waste haulage distance and an average cycle time (round trip) using the speed parameters shown in Table 16-48.

Early in the mine life, the average one-way ore haulage distances are approximately 2,500 m and increase to approximately 3,200 m. Waste haulage distances vary considerably depending on the time period. At the peak, five haul trucks are required to transport the ore, waste and CRF. Figure 16-36 and Figure 16-37 show the haulage distance and cycle time by period.



Source: SRK

Figure 16-36: Haulage Distance – One Way Length



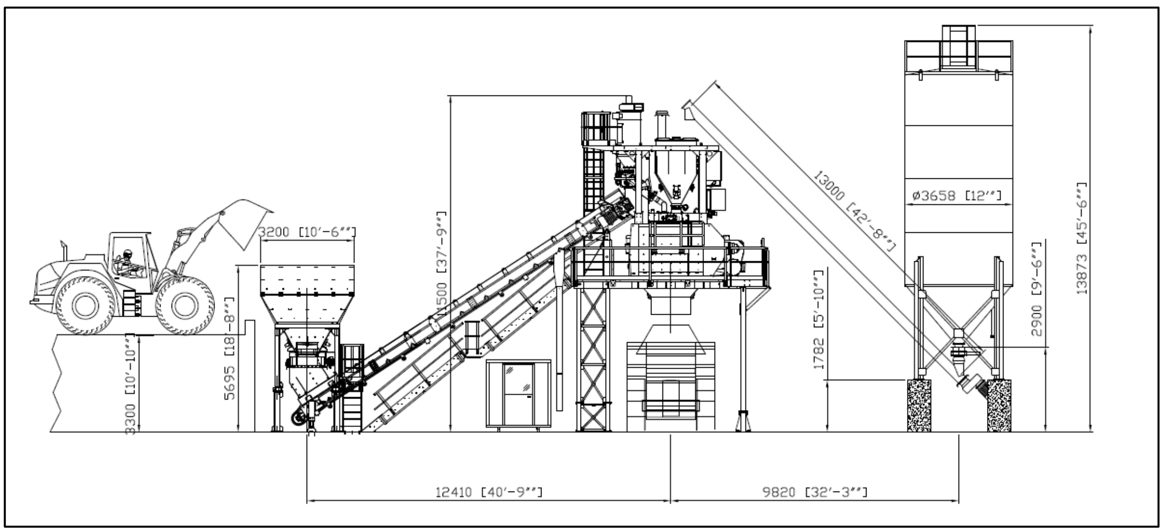
Source: SRK

Figure 16-37: Haulage Cycle Time – Roundtrip

Backfilling

The mine will utilize CRF in the primary stopes and either rock fill or low strength CRF in the secondary stopes. CRF will be generated in a surface plant located at the underground storage yard that includes a 160 t/hr portable crushing/screen plant that will create two specification grade aggregate piles and an oversize pile. The specification grade aggregate will be transported to the CRF plant by a front-end loader where it will be loaded into one of two hoppers, a large aggregate (4 inch to 3/16 inch) and fine aggregate (3/16 inch minus), that will in turn batch feed into a mixer that combines specified quantities of cement, water, and aggregate to create the required high strength CRF (4% cement on a dry basis, as discussed in section 16.2.2). A conveyor moves the CRF mixture to a bin that stores the CRF for loading of the underground trucks. The CRF plant has a capacity of 100 m^3/hr with a batch mixer, cement silo with screw conveyor and weigh hopper, water weigh hopper, and the aforementioned two loss-in-weight aggregate bins. The 40-t underground haul truck pulls under the bin after dumping its ore on the stockpile and loads the CRF. Once loaded, the truck hauls the CRF underground to an open stope where backfilling is taking place. The truck dumps the load either directly into the stope or in a staging area where an LHD hauls the CRF to the stope for placement.

Figure 16-38 shows the CRF plant.



Source: SIMEM, 2017

Figure 16-38: CRF Backfill Plant

The cement will be provided by a local manufacturer that will truck the cement in 25-t tanker transports to the site at the rate of approximately two trucks per week. Table 16-49 shows the total LoM volume breakdown of the rock fill (low strength) and CRF (high strength).

Table 16-49: LoM Backfill Quantities of Rockfill (low strength) and CRF (high strength)

Backfill Summary	(m^3)
Total Backfill Volume	**1,085,086**
Non-cemented waste rock	270,406
CRF (4% cement)	814,681

Source: SRK

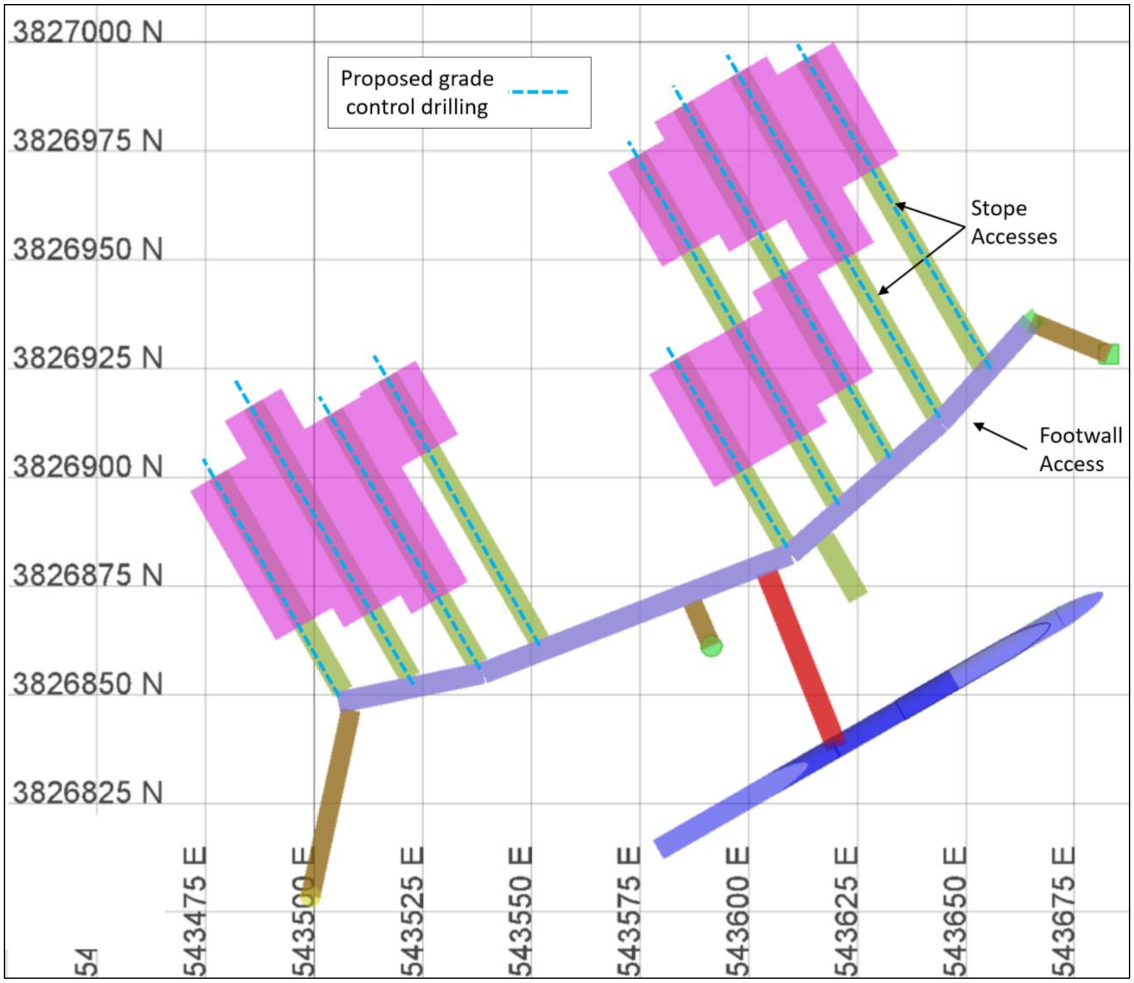
Waste Rock

Waste rock from the underground will be used as non-cemented fill in secondary stopes when possible. If a secondary stope is not available at the time the waste is mined, it will be hauled to the surface and placed in the UG stockpiling area where it will subsequently be either hauled via open pit trucks to the appropriate waste rock stockpile or used to make CRF. The waste rock is hauled to the surface primarily during the pre-production period prior to stopes being developed.

Grade Control

The characterization of ore versus waste and further geochemical waste classification will be completed through diamond core drilling of the stope accesses prior to mining. Once the footwall level access is established, horizontal drill holes will be drilled 3 m beyond the planned length of the stope access as shown in Figure 16-39. The core will be logged, sampled and analyzed to provide grade control and geochemical waste classification information. Geologic and block models will be updated with this information and ore/waste grade boundaries will be pre-determined prior to mining the stope accesses. Areas considered to be waste will be characterized geochemically to determine which stockpile the material should be sent to. Geochemical sampling techniques for the underground mine will follow existing open pit sampling techniques. Initially, all stope accesses will be drilled and sampled to ensure adequate definition for each stope. As knowledge of the deposit is gained, there may be an opportunity to increase the spacing of the drill holes.

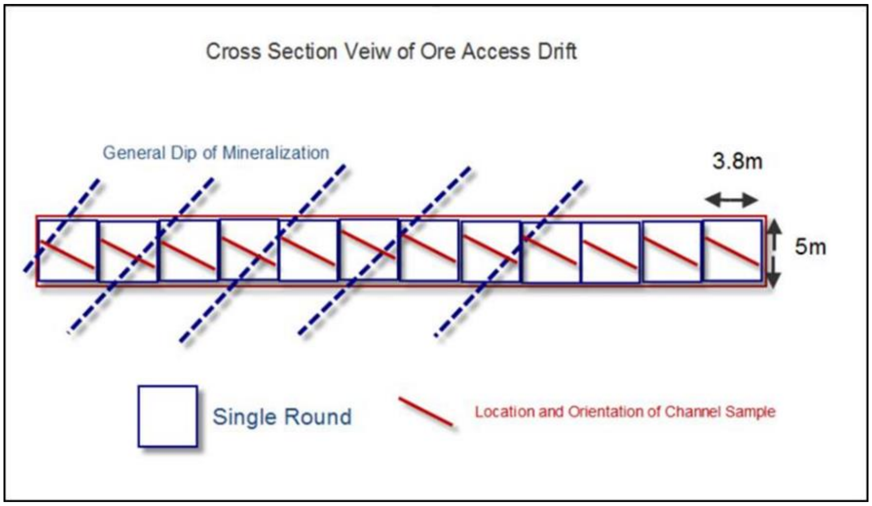
The existing onsite lab, Kershaw Minerals Lab (KML) analyzes Au, Ag, total carbon, and total sulfur. LECO is used to determine total carbon/sulfur on a percentage of production samples for detailed carbon and sulfur speciation. Lab turnaround time is expected to be 18 to 26 hours.



Source: SRK

Figure 16-39: Level Section Showing Planned Diamond Drilling from Footwall Access Drifts

Additionally, as the stope accesses are being mined, a mine geologist will observe, map and channel sample the exposed rock on each rib. Typically, the predominant fabric of the mineralization will display an inclined orientation on the ribs and the channel samples will need to be oriented normal to this fabric as shown in Figure 16-40.



Source: SRK

Figure 16-40: Cross Section View of Development Rounds and Proposed Channel Sampling Locations

The mine geologist will identify the presence of silicification or sulfidation and utilize these alterations to determine where sample breaks will occur. Using the inclined channel sample orientation, samples lengths for each cut will be 4 to 5 m in length. Samples will be collected as continuous channels at nominal 1 to 2 m lengths depending on the observations of the geologist. Channels can be sawn or chipped depending on the nature of the rock being collected. Sample breaks will be made at lithologic or alteration contacts.

Once mining has progressed to a point where sufficient testing has shown that estimated grades based on core samples reconcile to channel sampling during actual mining, a portion of the diamond drilling or channel sampling may be eliminated.

16.2.10 Ventilation

A ventilation system has been designed to support the development and production activities for the underground mine. The total life-of-mine analysis includes predicted distribution of airflow and pressure. The analysis is broken down into five phases, extending from the initial startup of construction activities to the completion of the decline and all associated levels and raises. A 3D ventilation model was created using Ventsim Visual 4.

<u>Input Parameters</u>

The location of the mine is in a very temperate area with the average low temperature in January of -1.3°C (average high 11.8°C), and an average high temperature in July of 32.6°C (average low 19.8°C). Combined with the shallow depth and apparent lack of geothermal activity, this mine does not appear to present temperature stress related issues, so studies in these areas were not developed.

No harmful strata gases are expected to be encountered at this site. No crushers or fixed ore/waste conveyances (continuous acute dust sources) are currently designed underground. Strategies for controlling dust while loading and hauling ore/waste are an important operational consideration with little impact at this design stage. The configuration of the system as an exhausting ventilation system which minimizes the blast clearance time/possibility of exposure to blast-generated gases by maintaining the ramp clear of blasting fumes.

Airway dimensions are as per the mine design, with the main ramp being 5 x 5.5 m and the raises to surface at 5 m diameter. Oval equivalent duct of 1.65 m equivalent was modeled for the main decline developments. Model friction factors, resistances, shock losses, etc. were used based on available data and standard best practice.

Airflow Requirements

The expected equipment load was calculated based on the equipment list. A generic airflow dilution value of 0.08 m^3/s per kW power (125 ft^3 per minute/brake horsepower (cfm/bhp)) for diesel engines has been used for this study because the exact equipment has not been identified. Airflow for individual pieces of equipment in the mine will need to meet the requirements of CFR57.5067 which refers to the nameplate dilution values determined by MSHA/NIOSH testing, or meet the EPA requirements. Overall, the total airflow requirement for the mine is approximately 220 m^3/s.

During development, the mucking cycle utilizes the most DPM-intensive equipment set. This entails one to two haul trucks and an LHD, the airflow for which will be used as the requirement to support development. The airflow requirement per equipment piece for this task is as follows:

- One 40-t Haul Truck (generic) – 32.4 m^3/s dilution
- One 14-t LHD (generic) – 19.4 m^3/s dilution

A total volume of 52 m^3/s is required for a one haul truck, one LHD configuration. A volume of 84 m^3/s would be required to operate two haul trucks and an LHD in the same development heading.

Mine-wide ventilation system modeling was conducted with a “mining block-centered” approach, allocating 80 m^3/s (somewhat in excess of the 71.2 m^3/s required for two LHDs and one haul truck) to one stoping area. This 80 m^3/s was divided equally between the upper and lower accesses to a given stope (40 m^3/s to each), representing a longer-term steady-state of mucking and backfilling stopes.

During stope access development, as well as production, one LHD is assumed to be operating in a stope access at a time. The volume required for one LHD, 19.4 m^3/s, is needed for each stope for most of the mining cycle. During backfill, one haul truck may access a stope at a time to directly dump backfill into a stope, so a total volume of 32.4 m^3/s will be needed for each stope access during backfill.

Auxiliary Ventilation Systems

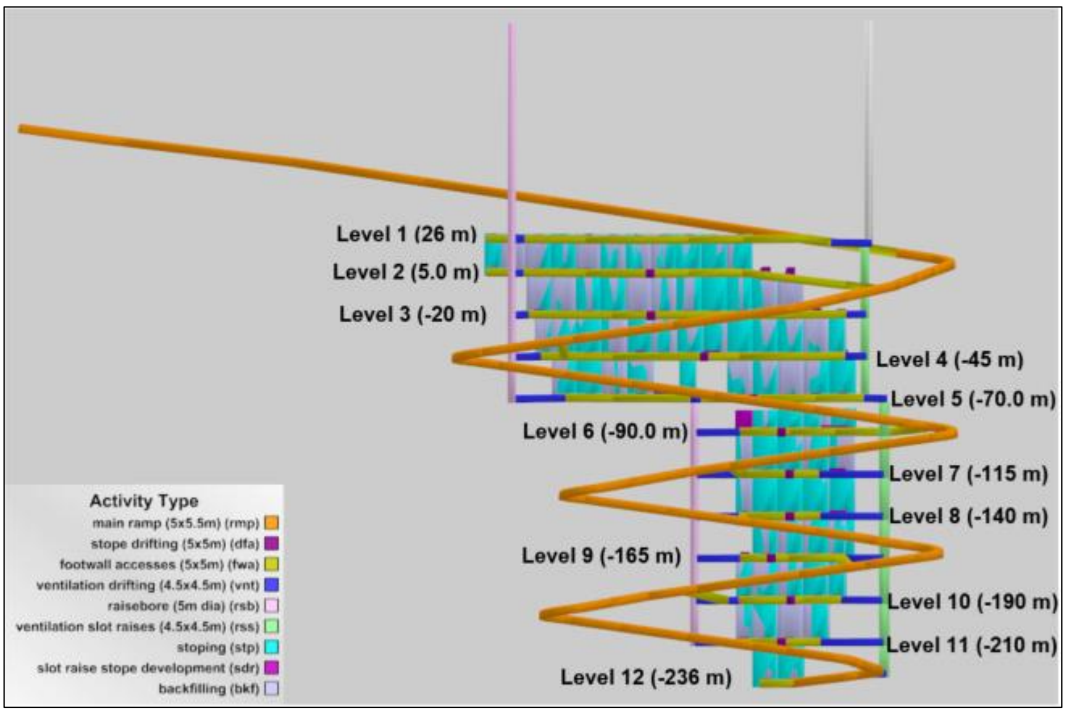
The decline development auxiliary ventilation system is sized to ventilate for the simultaneous operation of one truck and one LHD. The decline auxiliary ventilation system will consist of:

- One, two, or three fans operating in series that can be added as the length of the development increases. A single fan will be able to be used until the length (resistance) of the duct increases to a point where the airflow delivered to the face is impacted, at which time the second fan should be added and so forth.
- The single 1.65 m oval equivalent duct can be used to develop the initial decline. In the immediate development face area, a length of supplemental duct can be attached to the main duct to provide ventilation directly at the face during blasting, and during loading. The supplemental duct is advanced with the face and is not left incorporated within the long-term duct installation.

For the stope auxiliary ventilation system, auxiliary fans were selected to provide the 19.4 m^3/s required for development or production in a stope. A maximum duct length of 120 m and 1.2 m diameter flexible duct was assumed for this specification for a forcing auxiliary system. If a haul truck will enter the stope access during the backfill cycle, a second duct system was assumed to be installed in parallel with the first, to provide approximately 39 m^3/s – more than sufficient for a single haul truck. Up to four stopes were assumed to be active at once: either in development, production, or backfilling. Each stope will require two auxiliary fan/duct systems, with one fan/duct system moved up from the mucking level to the backfill level after each stope completes production.

<u>Ventilation Model</u>

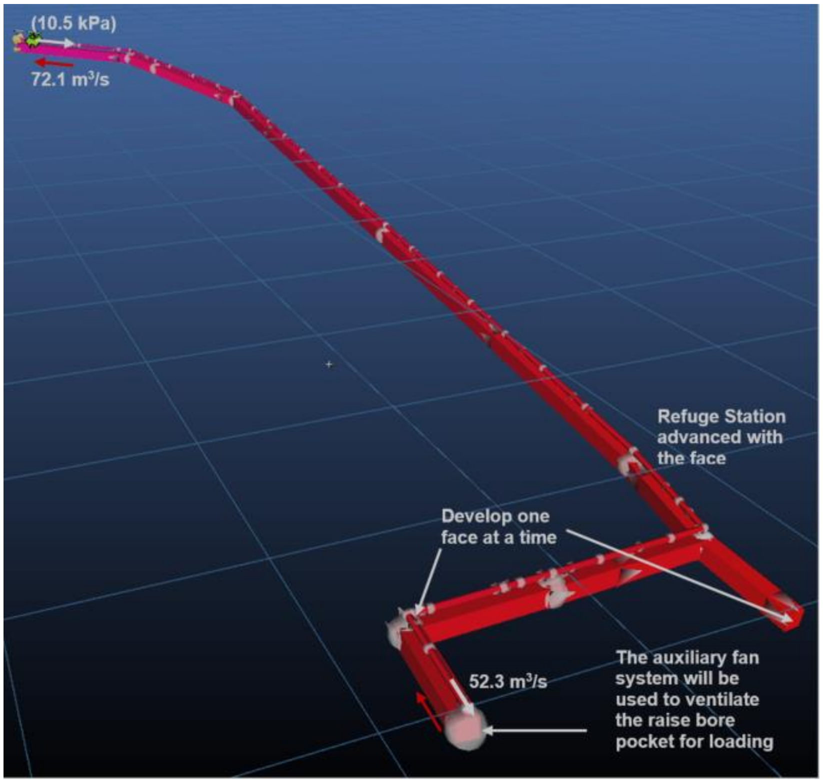
Model work was completed using 5 stages. Level naming nomenclature is shown in Figure 16-41.



Source: SRK

Figure 16-41: Ventilation level nomenclature

Stage 1 (August 2019 to February 2020) is the initial development of the decline and exhaust raise. As the decline continues onto the first level, the duct will extend with the development up to the 730-m length of ducting required to ventilate the raise boring of the exhaust raise, as shown in Figure 16-42.

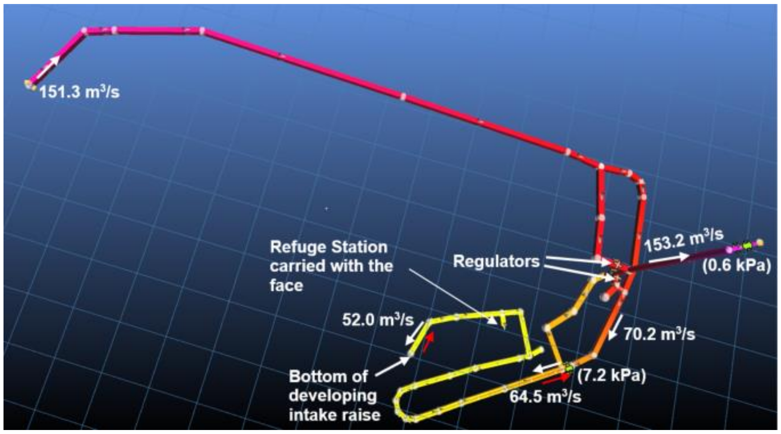


Source: SRK

Figure 16-42: Stage 1 airflow distribution

The auxiliary fan system operates at 10.5 kPa and uses two auxiliary fans in series and a third fan in series hung in the decline downstream of the two fans. The two-fan system will be positioned outside of the portal. As development of the decline progresses, the ducting will be extended to the new face as needed and the third fan will be added as airflow demands increase. An initial quantity of 72.1 m^3/s enters the ducting and 52.3 m^3/s is delivered to the raisebore pocket. This quantity should be sufficient to support one truck and one LHD as the decline and initial raise access are developed.

Stage 2 (February 2020-August 2020) includes further development of the decline, level development along this section of decline, and the eventual addition of the intake raise as shown in Figure 16-43. The fan to be installed at the top of the exhaust raise should begin operation at a duty of approximately 155 m^3/s at 0.6 kPa and operate at approximately this point for the entirety of Stage 2. This will establish the initial flow-through ventilation in the mine.



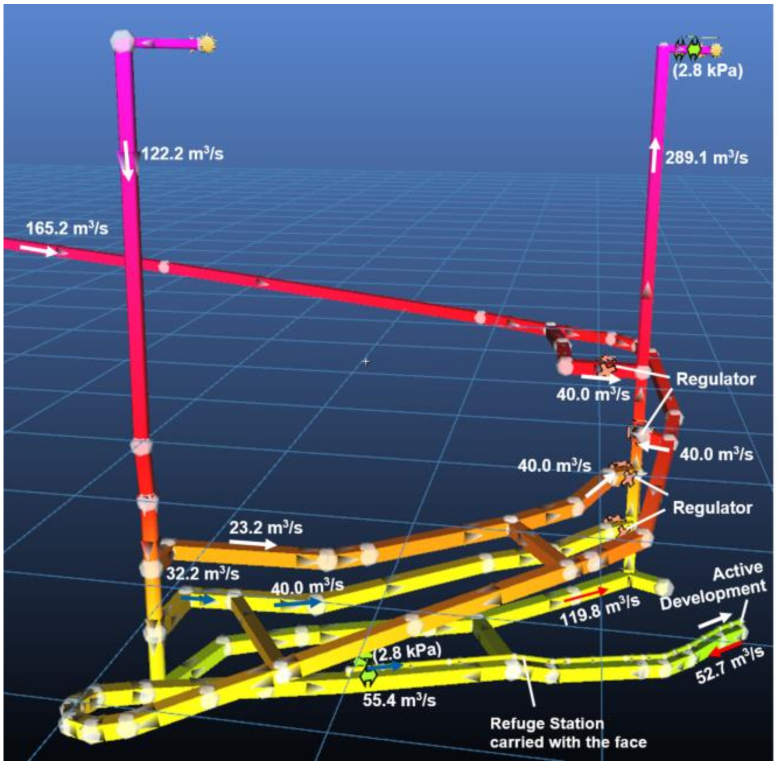
Source: SRK

Figure 16-43: Stage 2 airflow distribution

The decline development is highly dependent on the continued development of levels. Each level connects into the exhaust raise via slot raises. Once a lower level is properly connected into the exhaust system, a regulator is placed on the exhaust access airway of the level above it. These regulators are set to limit airflow to 40 m^3/s. This method will effectively draw more airflow deeper into the mine, through the new level and out to the exhaust slot raise. The availability of flow-through airflow farther down the decline will allow the auxiliary fan system and ducting to advance down ramp, closer to the working face. This “leapfrogging” process shortens the necessary ducting length and utilizes fan power more efficiently.

The development in Stage 2 will progress to develop levels 1, 2, and 3 from their decline accesses to the exhaust airways. Level 5 is developed from its decline access to the bottom of the planned intake raise. As shown in Figure 16-44, a fan pressure of 7.2 kPa moves a total of 64.5 m^3/s along 555 m of ducting. A quantity of 52.0 m^3/s is delivered to the decline development activity.

Stage 3 (August 2020 – September 2021) entails finishing construction on the intake raise, further development of levels and decline advancement. Upon the completion of the intake raise at the end of Stage 2, the overall mine airflow can be increased. The exhaust fan pressure is elevated to 2.8 kPa, raising the overall airflow to 289.1 m^3/s. Levels 3, 4, and 5 are developed to connect into both the intake and exhaust raises. Regulators are placed on the Level 3 and Level 4 exhaust connections to limit the level airflow to 40.0 m^3/s each. This stage also includes the large-scale development and production of the stopes between Levels 4 and 5. Stage 3 airflow distribution is shown in Figure 16-44.

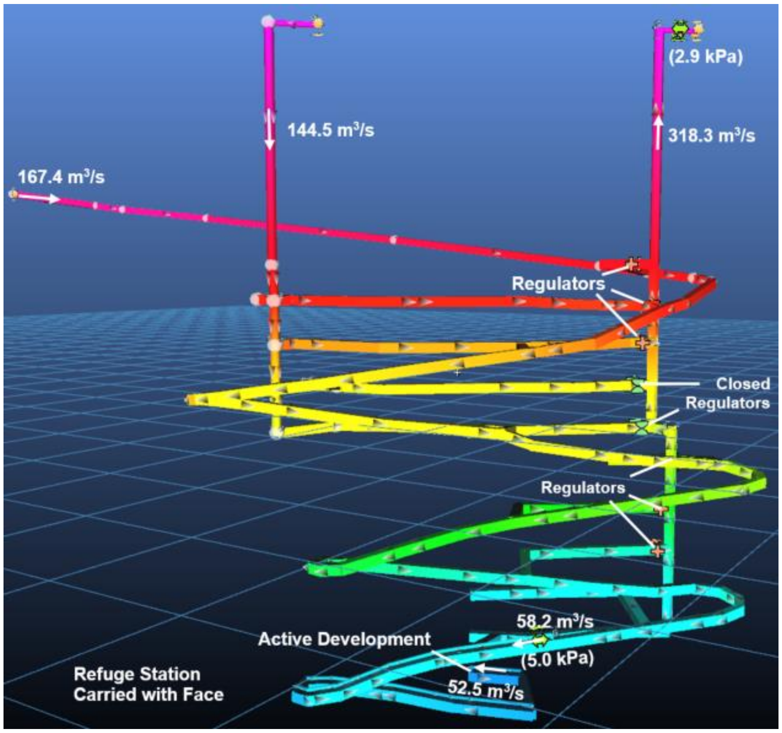


Source: SRK

Figure 16-44: Stage 3 airflow distribution

An auxiliary fan pressure of 2.8 kPa moves a total of 55.4 m^3/s along 206 m of ducting. A quantity of 52.7 m^3/s is delivered to the development face.

Stage 4 (September 2021-February 2023) is the development of the lower portion of Haile leading up to the boring of the internal raise. This stage includes the construction of the main decline, portions of the lower levels, and the exhaust slot raises. The lower mine levels constructed during this stage will be connected into the exhaust airway via slot raises. Regulators will be used to distribute the airflow between the levels. Stage 4 concludes with the auxiliary system ventilating the starting point of the raise bore for the development of the internal intake raise. Stage 4 air distribution is shown in Figure 16-49.

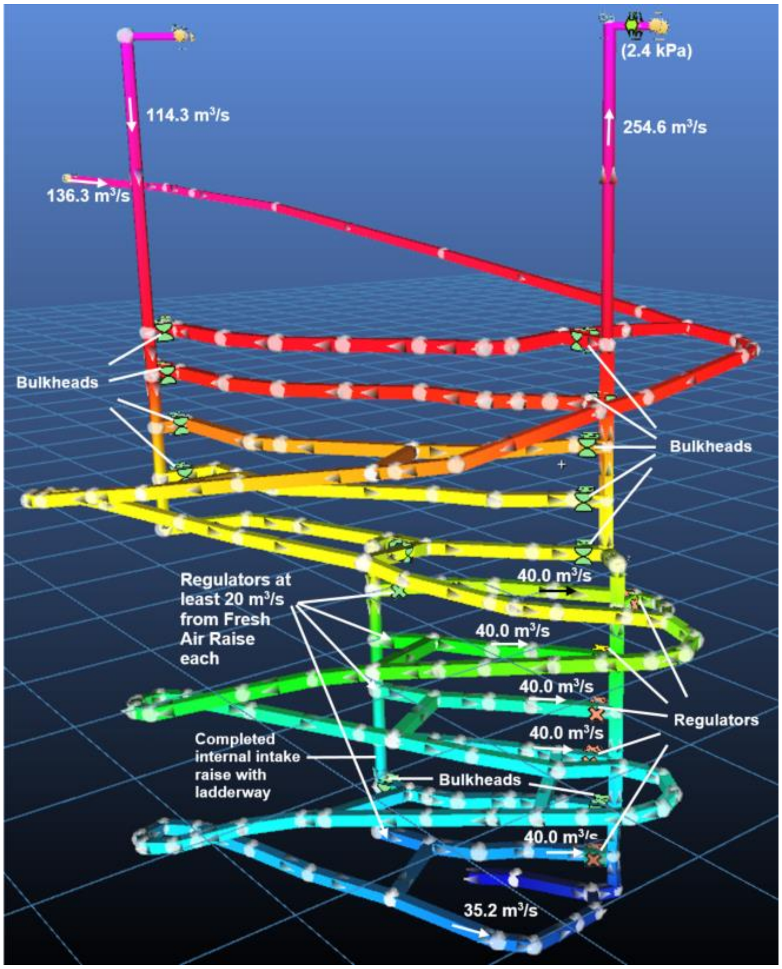


Source: SRK

Figure 16-45: Stage 4 Ventilation Schematic

An auxiliary fan pressure of 5.0 kPa moves a total of 58.2 m^3/s along 206.4 m of ducting. This fan pressure requires only one of the auxiliary fans to provide the necessary airflow. A quantity of 52.5 m^3/s is delivered to the development face.

Stage 5 (February 2023 to end of mine life) entails the commisioning of the internal intake raise, completion of the planned levels, and beginning of full-scale production from the lower block. The internal intake raise will carry 100 to 200 m^3/s from the intake raise connected to surface. The Stage 5 airflow distribution is shown in Figure 16-46.



Source: SRK

Figure 16-46: Stage 5 Ventilation Schematic

In Stage 5, most of the mine's upper levels and stopes have been mined out and the mine airflow is directed to the lower portion of the mine with the use of ventilation controls. Stage 5 overall airflow quantity will be 250 m^3/s. A quantity of 110 m^3/s will enter the mine at the intake raise to surface and a quantity of 140 m^3/s will enter the mine through the main portal. The maximum air velocity in the ramp will occur just beyond the portal. At the end of Stage 5, with completed development, this velocity value for the decline will be 5 m/s.

16.2.11 Mine Services

Dewatering

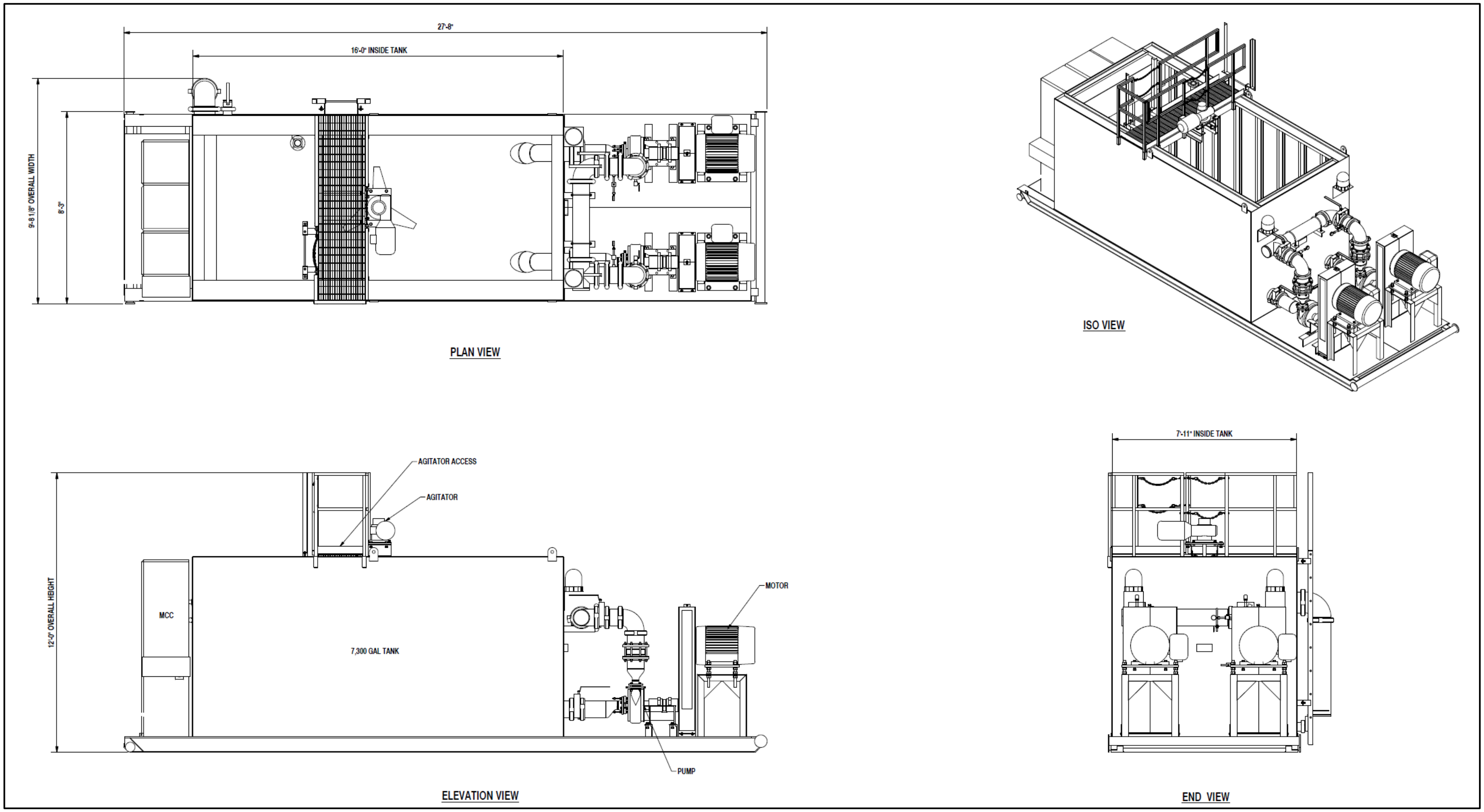
Based on the water inflow work discussed in section 16.2.3, a 63.1 L/s (1,000 gpm) capacity, 31.5 L/s (500 gpm) operating, dewatering system was designed.

The larger peak inflows are expected to be encountered in the decline near surface. The weathered metavolcanics in this area have an enhanced permeability as compared to the other units. Once through the weathered zone, lower amounts of water are anticipated.

The system is built in phases as the mine develops and consists of a portable development system and permanent level pump stations that will be constructed above the sill levels as the mine production levels are constructed. The system will pump from the underground workings through steel and SDR11 8-inch HDPE pipes to an existing lined surface pond (Pond 465) where the water will be returned through an existing system to the existing water treatment plant at the processing plant.

The portable development system will be used from the portal to level -70. Once the permanent pump system is completed on the -70 level and piping out through the vent hole is completed, the portable system will be removed from the decline and utilized for the development decline progressing to access level -240. A second permanent pump station will be established on the -240 level that will pump from the -240 level to the -70 level, where the -70 level pumps will pump the water to the surface for discharge to Pond 465.

The portable development system will consist of four complete pump skid stations that provide 62 liters per second (L/sec) (1,000 gpm) at 59 m each with a 28.4 m^3 capacity tank with electrical motor control center (MCC) mounted on the skid, as shown in Figure 16-47.



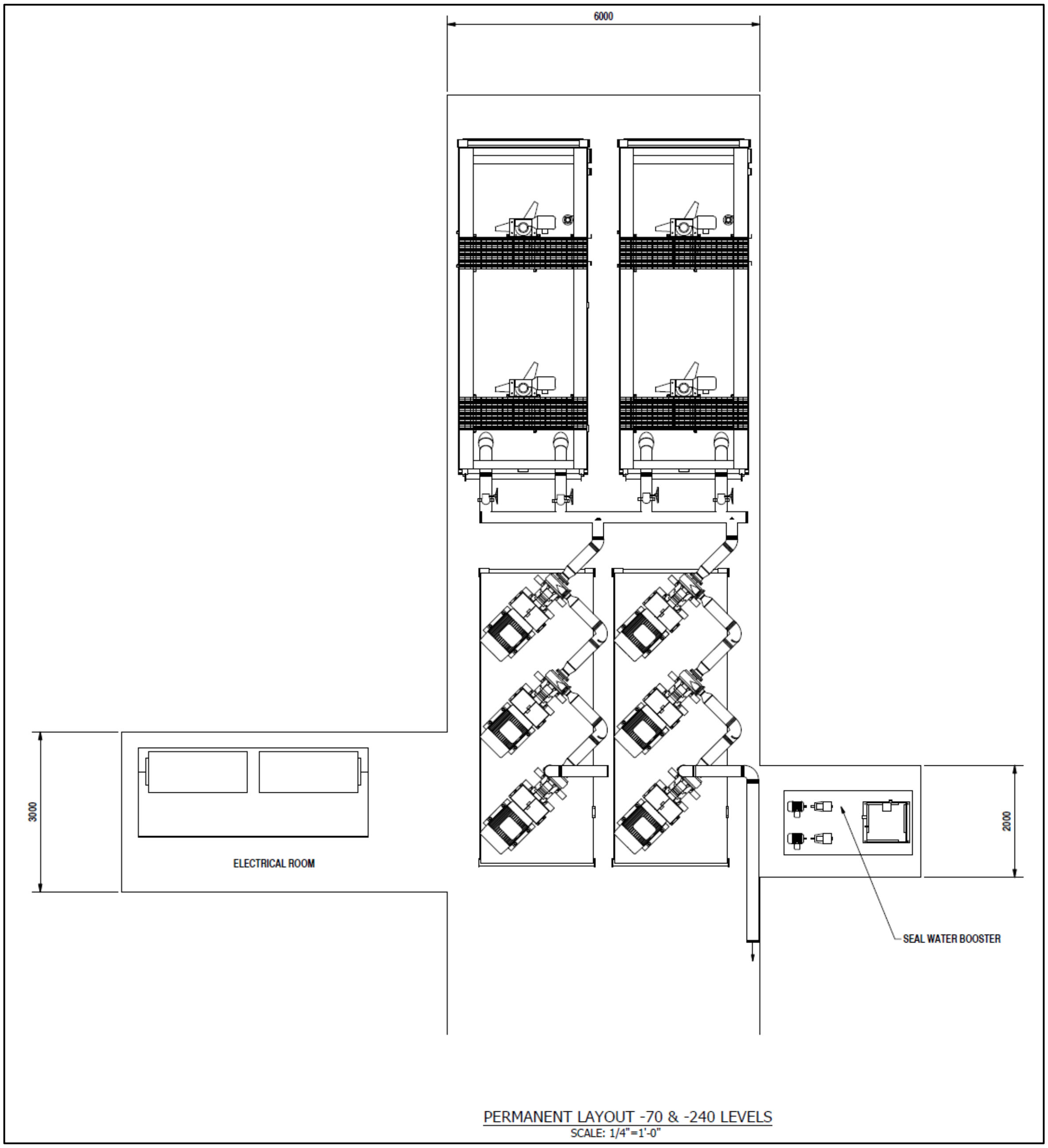
Source: Miller Engineering, 2017 (units shown in ft/inches)

Figure 16-47: Portable Pumping Skids

The -70 level permanent pumping station will consist of two sets (one operating/one on standby) of three mounted 150 HP pumps in series to provide 66 L/s at 238 TDH with two agitated steel 7,500 gallon tanks, water hammer surge protection, and MCC. The entire system will be skid mounted.

The -240 level permanent pumping station will consist of two sets (one operating/one on standby) of three 100 HP pumps in series to provide 66 ls/s @ 174 TDH with two agitated steel 7,500 gallon tanks, water hammer surge protection and MCC. The full system is skid mounted.

Figure 16-48 shows the permanent pumping station configuration.



Source: Miller Engineering, 2017

Figure 16-48: Permanent Pumping Skids

Surface dewatering will be ongoing during the time of underground production and a surface dewatering well is anticipated to be located near the main decline to minimize water encountered during development through the weathered zone.

Electrical

The mine electrical system will be supplied from the existing power line from the main substation at the processing plant through the 24.9kV power line that feeds Pond 465. The line will be further developed to the underground yard and eventually to the ventilation raise to feed the mine.

Initially power will come from the underground yard through a feeder cable to the portable mine substation located at the portal location that will provide power for the portal fans used during development, mine pump system, mine equipment used for driving the access decline, and auxiliary power for miscellaneous equipment required during development of the mine. Power will be carried down the decline in power cables suspended in the back to portable subs that will step the power down to feed the mobile equipment and pumps.

Once the mine ventilation hole is established, a new 13.8 kV overhead line from the underground storage yard will be extended to the vent hole and a main power feed will be run down the vent raise to feed the mine through a distribution system at 13.8kV to portable substations that feed the various mine ventilation, mobile, fixed, and pumping equipment.

The underground mine connected load including all equipment at full capacity is approximately 6 MW. The LoM average usage is 1.2 million kWhr per month with an average operating load of 2.4 MW

Health & Safety

The mine design incorporates MSHA safety standards and includes emergency hoists in the two raisebored fresh air intake raises. The hoists will be connected to backup power generation. Additionally, 12-person and 8-person mobile refuge chambers are included and will be located in active working areas over the LoM.

The mine will have a communications system that has both mine phones and wireless communication through a leaky feeder system. A mine rescue team will support the operation. The mine safety program will integrate with local providers in case of any mine emergency. A stench gas emergency warning system will be installed in the mine's intake ventilation system. This system can be activated to warn underground employees of a fire situation or other emergency whereupon emergency procedures will be followed.

Manpower

Manpower levels are estimated based on the production schedule and associated equipment operating requirements. The estimate is based on owner mining using a mine operating schedule consisting of 12 hours per shift, two shifts per day, and 7 days/week. Each 12-hr shift is supported by a four-crew rotation working a 4 x 3 roster (i.e., 26 weeks of the year the crew will work 4 days and 26 weeks of the year the crew will work 3 days). The 4 x 3 roster results in 1,976 hours per year being worked at regular wage rate and 208 hours per year being worked at the overtime wage rate (i.e., time and a half).

The management and technical team will work five 8-hr days per week.

Table 16-50 shows the required workforce.

Table 16-50: Typical Mining Labor by Shift

Day Shift	Qty	
Mining Manager	1	
Mine Superintendent	1	
Maintenance Superintendent	1	
Technical Services Superintendent	1	
Safety Supervisor	1	
Mine Clerk	1	
Chief Engineer	1	
Geotechnical Engineer	1	
Long Term Mine Planner	1	
Short Term Mine Planner	2	
Project Engineer (vent, water, construction)	1	
Chief Geologist	1	
Resource Geologist	1	
Grade Control Geologist	1	
Chief Surveyor	1	
Mine Surveyor	2	
Total	**18**	
Rotating Shift	**Per Shift (Qty)**	**Total (Qty)**
Safety Technician / Trainer	1	4
Shift Supervisor	1	4
Blasting/Powder Crew - Miner II	2	8
Blasting/Powder Crew Helper - Miner III	2	8
Jumbo Operator - Miner I	2	8
Longhole Drill Operator - Miner I	1	4
DTH Drill Operator - Miner I	1	4
Scooptram Operator - Miner II	2	8
Scooptram Operator - Miner III	2	8
Haul Truck Operator - Miner II	2	8
Haul Truck Operator - Miner III	3	12
Bolter Operator - Miner I	1	4
Bolter Operator - Miner II	1	4
Cable Bolter Operator - Miner I	1	4
Nipper - Miner III (also vacation coverage)	4	16
Mine Services Shift Supervisor	1	4
Utility Crew - Miner II	2	8
Utility Crew - Miner III	3	12
Mine Electrician -Miner I	1	4
Diamond Driller	1	4
Maintenance Supervisor	1	4
Heavy Equipment Mechanic I	2	8
Heavy Equipment Mechanic II	3	12
Heavy Equipment Mechanic III	4	16
Electric/Hydraulic Mechanic I	2	8
Electric/Hydraulic Mechanic II	1	4
Total	**47**	**188**

Source: SRK

The estimated labor rates are consistent with the existing open pit wage scale; however, one additional wage category was added to allow for a limited number of highly skilled and experienced workers in the Miner I and Mechanic I categories (e.g., jumbo operators).

Equipment

The underground mobile equipment requirements as summarized in Table 16-51 are based on the production schedule and an assumed LoM mechanical availability of 85%.

The yearly fixed equipment list is shown in Table 16-52.

Table 16-51: Mobile Equipment in Operation, by Year

Type of Equipment	Diesel (kW)	Electric (kW)	2018	2019	2020	2021	2022	2023	2024	2025	Maximum
DTH	119	119			1	1	1	1	1	1	1
TH Drill	110	119			1	1	1	1	1	1	1
Explosives Charger (large)	158				1	1	1	1	1	1	1
LHD's (14T)	256			1	1	2	2	2	2	2	2
Jumbos	119	200		1	2	2	2	2	2	2	2
Explosive Chargers (small)				1	1	1	1	1	1	1	1
Bolters	70	70		1	2	2	2	2	2	2	2
Scissor Lifts	127			1	1	1	1	1	1	1	1
Trucks (40T)	405			1	2	5	5	5	5	5	5
Cable Bolter	120	63		-	1	1	1	1	1	1	1
LHD (7T) (includes placing CRF in stopes)	150			1	2	2	2	2	2	2	2
Boom Truck	127			1	1	1	1	1	1	1	1
Lube Truck	127			1	1	1	1	1	1	1	1
Fuel Truck	127			1	1	1	1	1	1	1	1
Flat Bed (for explosives chargers)	170			1	2	2	2	2	2	2	2
Shotcrete Sprayer Truck	170			1	1	1	1	1	1	1	1
Transmixer Truck	170			1	1	1	1	1	1	1	1
Telehandler	106			1	1	1	1	1	1	1	1
Skid Steer	73			1	1	1	1	1	1	1	1
Personnel Carriers (RTV style, four person)	19			2	4	4	4	4	4	4	4
Personnel Carrier (16 person)	127				2	2	2	2	2	2	2
Underground Core Drill		55			1	1	1	1	1	1	1
Grader	114				1	1	1	1	1	1	1
Light Vehicles (4x4 Pickup Trucks)				6	6	6	6	6	6	6	6

Source: SRK

Table 16-52: Fixed Equipment List, by Year

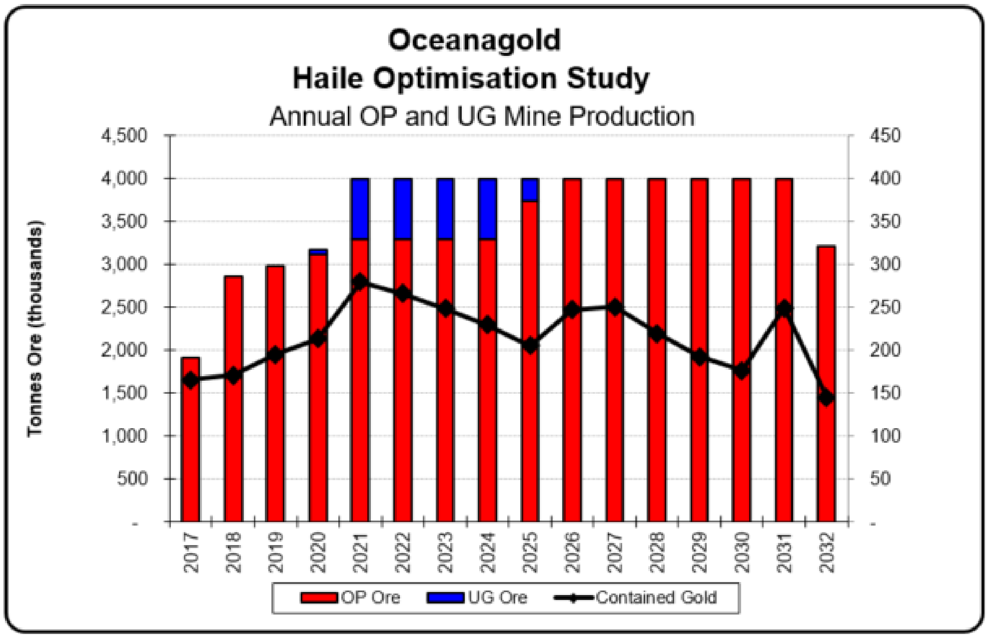
Type of Equipment	Diesel (kW)	Electric (kW)	2018	2019	2020	2021	2022	2023	2024	2025	Total
CRF Plant - CRF Plant: 3.5m^3 batch / 100 m^3/hr					1.00						1
Mobile Screen Plant					1.00						1
Water Tank for Plant					1.00						1
Power Line and Transformer for CRF Plant					1.00						1
Preproduction Fan System (two each): 250 hp/186 kW)		186	-	1.00							1
Move existing mobile truck shop to UG yard location				1.00							1
Shipping Container Storage				2.00							2
Engineering Equipment Allowance					1.00						1
Emergency Hoist (plus installation)					1.00			1.00			2
Powder/Primer totes for transport (powder 3,400 lb each) (need 13 per Orica)				6.00	7.00						13
Stationary mine substation 5,000 kVa				1.00							1
Portable Power Center 500 kVA (for mobile equipment)				2.00	3.00						5
Portable Power Center 1500 kVA (for pump stations – use two for portal)				2.00							2
Main power line to UG yard (estimate 1km)				1.00							1
Main substation at UG yard				1.00							1
Backup Generator for ventilation system				1.00							1
Dry and Office Building				1.00							1
Tool Allowance for Shop				1.00							1
Refuge Chambers (12 person)						1.00					1
Refuge Chambers (8 person)				1.00							1
Mobile 200 amp welders				1.00							1
Mobile 400 amp welder				1.00							1
Communications Infrastructure and Mine automation				0.50	0.50						1
Service road to the vent hole				1.00							1
Portal Construction (contracted cost used)				1.00							1
Main Fan System at vent hole (1500 HP/1117.5 kW) with starter/VFD						1.00					1
Stope Development Fans (100 HP/745 kW)					1.00						1
Staged Pump system for Decline and Ramps and discharge to surface pond				0.50	0.75	0.25					2
Development Stage Secondary Submersible Pumps				0.50	0.50						1
Pump System for level -70 (redundant 150 HP pumps 6/3 operating)					1.00						1
Pump System for level -240 (redundant 150 HP pumps 6/3 operating)								1.00			1
Mobile Crusher - Jaw Crusher (160 t/hr -72 mm)					1.00						1

Type of Equipment	Diesel (kW)	Electric (kW)	2018	2019	2020	2021	2022	2023	2024	2025	Total
Bulkheads - Cost Each							2.00	9.00			11
Regulator - Cost Each					2.00	2.00	3.00	9.00			16
Air door - Cost each								2.00			2
Piping - 8” Steel Sch40 grooved w/coupling				0.55	0.45						1
-70 level - Discharge Piping – 8-inch Sch40 Steel grooved end w/coupling 2020					1.00						1
-240 level - Discharge Piping – 8-inch Sch40 Steel grooved end w/coupling 2020								1.00			1

Source: SRK

16.3 Combined Open Pit and Underground Production Schedule

Figure 16-49 and Table 16-53 and show the combined open pit and underground production schedule annually.



Source: SRK

Figure 16-49: Combined Open Pit and Underground Production

Table 16-53: Combined Open Pit and Underground Production Schedule

Year		2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
Total Production												
Ore	kt	1,921	2,862	2,984	3,170	4,000	4,000	4,000	4,000	4,000	4,000	4,000
Ore	t/d	5,262	7,841	8,176	8,660	10,959	10,959	10,959	10,959	10,959	10,959	10,959
Au Grade	g/t	2.84	2.07	2.06	2.03	2.19	2.06	1.92	1.77	1.58	1.96	1.95
Contained Au	g	5,463,211	5,913,835	6,148,709	6,423,141	8,755,490	8,252,009	7,687,379	7,099,306	6,314,599	7,820,260	7,813,751
Contained Au	oz	175,646	190,134	197,685	206,509	281,495	265,308	247,155	228,248	203,019	251,427	251,218
Underground												
Ore	kt	0	0	0	45	701	701	701	703	265	0	0
Ore	t/d	0	0	0	122	1,921	1,921	1,921	1,927	726	0	0
Au Grade	g/t	0	0	0	4.54	4.42	4.77	4.03	4.46	3.95	0	0
Contained Au	g	0	0	0	202,441	3,098,703	3,346,856	2,825,139	3,132,689	1,047,387	0	0
Waste	kt	0	0	48	181	96	117	114	29	-	0	0
Backfill Volume	kt	0	0	0	4	224	253	250	248	107	0	0
Open Pit												
Ore	kt	1,921	2,862	2,984	3,125	3,299	3,299	3,299	3,297	3,735	4,000	4,000
Ore	t/d	5,262	7,841	8,176	8,539	9,038	9,038	9,038	9,032	10,233	10,959	10,959
Au Grade	g/t	2.84	2.07	2.06	1.99	1.71	1.49	1.47	1.20	1.41	1.96	1.95
Contained Au	g	5,463,211	5,913,835	6,148,709	6,220,700	5,656,787	4,905,153	4,862,240	3,966,617	5,267,212	7,820,260	7,813,751
Strip ratio	t/t	9.3	6.8	13.7	13.1	11.1	10.8	8.9	11.1	9.7	9.0	9.0
Red Waste	kt	5,375	5,801	6,956	11,512	11,906	12,041	13,736	12,023	16,889	12,982	20,645
Yellow Waste	kt	1,912	2,018	2,513	3,341	27	143	(0)	947	(0)	(0)	(0)
Green Waste	kt	10,551	11,616	31,546	26,022	24,767	23,517	15,666	23,733	19,376	23,018	15,355
Waste Total	**kt**	**17,837**	**19,435**	**41,016**	**40,875**	**36,701**	**35,701**	**29,401**	**36,703**	**36,265**	**36,000**	**36,000**

Year		2028	2029	2030	2031	2032	Total
Total Production							
Ore	kt	4,000	4,000	4,000	4,000	3,218	58,155
Ore	t/d	10,959	10,959	10,959	10,959	8,817	10,959
Au Grade	g/t	1.68	1.48	1.36	1.99	1.18	1.85
Contained Au	g	6,736,256	5,917,892	5,427,136	7,961,701	3,793,881	107,528,556
Contained Au	oz	216,575	190,265	174,486	255,974	121,976	3,457,121
Underground							
Ore	kt	0	0	0	0	0	3,116
Ore	t/d	0	0	0	0	0	1,921
Au Grade	g/t	0	0	0	0	0	4.38
Contained Au	g	0	0	0	0	0	13,653,215
Waste	kt	0	0	0	0	0	584
Backfill Volume	kt	0	0	0	0	0	1,085
Open Pit							
Ore	kt	4,000	4,000	4,000	4,000	3,218	55,039
Ore	t/d	10,959	10,959	10,959	10,959	8,817	9,038
Au Grade	g/t	1.68	1.48	1.36	1.99	1.18	1.71
Contained Au	g	6,736,256	5,917,892	5,427,136	7,961,701	3,793,881	93,875,341
Strip ratio	t/t	9.0	8.2	5.8	2.2	4.5	8.7
Red Waste	kt	28,474	24,558	19,206	7,648	9,339	219,090
Yellow Waste	kt	(0)	(0)	(0)	0	0	10,901
Green Waste	kt	7,526	8,442	3,794	994	5,259	251,183
Waste Total	**kt**	**36,000**	**33,000**	**23,000**	**8,642**	**14,598**	**481,174**

Source: SRK

17 Recovery Methods

A conventional flotation and cyanide leaching flow sheet is used at HGM. The following summarizes the current operating process used to extract gold and silver.

The process plant consists of the following major components:

- Crushing and conveying;
- Storage and stockpiling of ore and reclaim;
- Grinding;
- Flotation;
- Regrinding of concentrate;
- Carbon in leach (CIL) recovery of precious metal values from reground flotation concentrate and flotation tailings;
- Acid washing and elution of precious metal values from CIL loaded carbon;
- Electrowinning and refining of precious metal value;
- Thermal regeneration of eluted and carbon and recycle to CIL; and
- CIL tailing thickening, cyanide recovery, detoxification and pumping of slurry to storage.

The following section describes the plant operation currently in operation at Haile.

A relatively compact RoM ore area is provided for either direct tipping of ore deliveries, or storage and re-handling, into the crusher.

An apron grizzly feeder onto a vibrating grizzly that delivers scalped oversize to the primary jaw crusher to reduce the ore size from RoM to minus six inches. Crushed ore is conveyed for surge and storage of the recombined grizzly undersize fines and primary crushed ore in a coarse ore surge bin or diverted on to an open conical emergency stockpile for later reclaim by Front End Loader (FEL) into a reclaim bin.

Ore is reclaimed from either the surge or reclaim bins, separately or simultaneously, using apron feeders on to a SAG mill feed conveyor belt delivering into the SAG mill feed chute.

Ore is milled in the SAG–Ball mill circuit. The SAG mill operates in closed circuit with a vibrating discharge screen and a pebble return circuit. The ball mill operates in closed circuit with hydrocyclones to produce the desired grinding product size.

Selected flotation reagents are added in the grinding circuit and a portion of the grinding circuit ball mill circulating load is treated in a flash flotation cell with the concentrate going to the regrind circuit.

The grinding circuit product passes to a bank of bulk rougher flotation cells to recover the balance of the sulfide mineralization. Thereafter the combined flash and rougher flotation concentrates are reground in stirred mills.

The reground concentrate slurry is dewatered in a high-rate thickener prior to transfer to a tank for the pre-aeration step followed by cyanide leaching in a Carbon-in-Leach circuit to dissolve gold and silver and adsorb the precious metal values from the solution onto activated carbon.

The flotation tailings slurry is thickened to recycle process water to the grinding circuit and the thickened tails slurry will be combined with the leached concentrate stream and processed in an extension of the carbon in leach circuit to recovery any leachable gold and silver contained in the float tail.

The loaded carbon is removed via screens from the CIL circuit and after further treatment by acid washing to reduce calcium scaling, precious metals are stripped with hot caustic-cyanide solution. The gold and silver are recovered by electrowinning from this solution and the stripped carbon is heated in a kiln under a reducing atmosphere for thermal reactivation of its adsorption capability before being returned to the CIL tanks for reuse.

The precious metal sludge from the electrowinning cells is dried and blended with fluxes and smelted to produce gold-silver doré bars, which are the final product of the ore processing facility.

The tailing slurry exiting the CIL is dewatered in a similar thickening stage to the flotation tailings for recovery of the cyanide solution and reduction of the volume of slurry needing to be treated by oxidation of the residual cyanide.

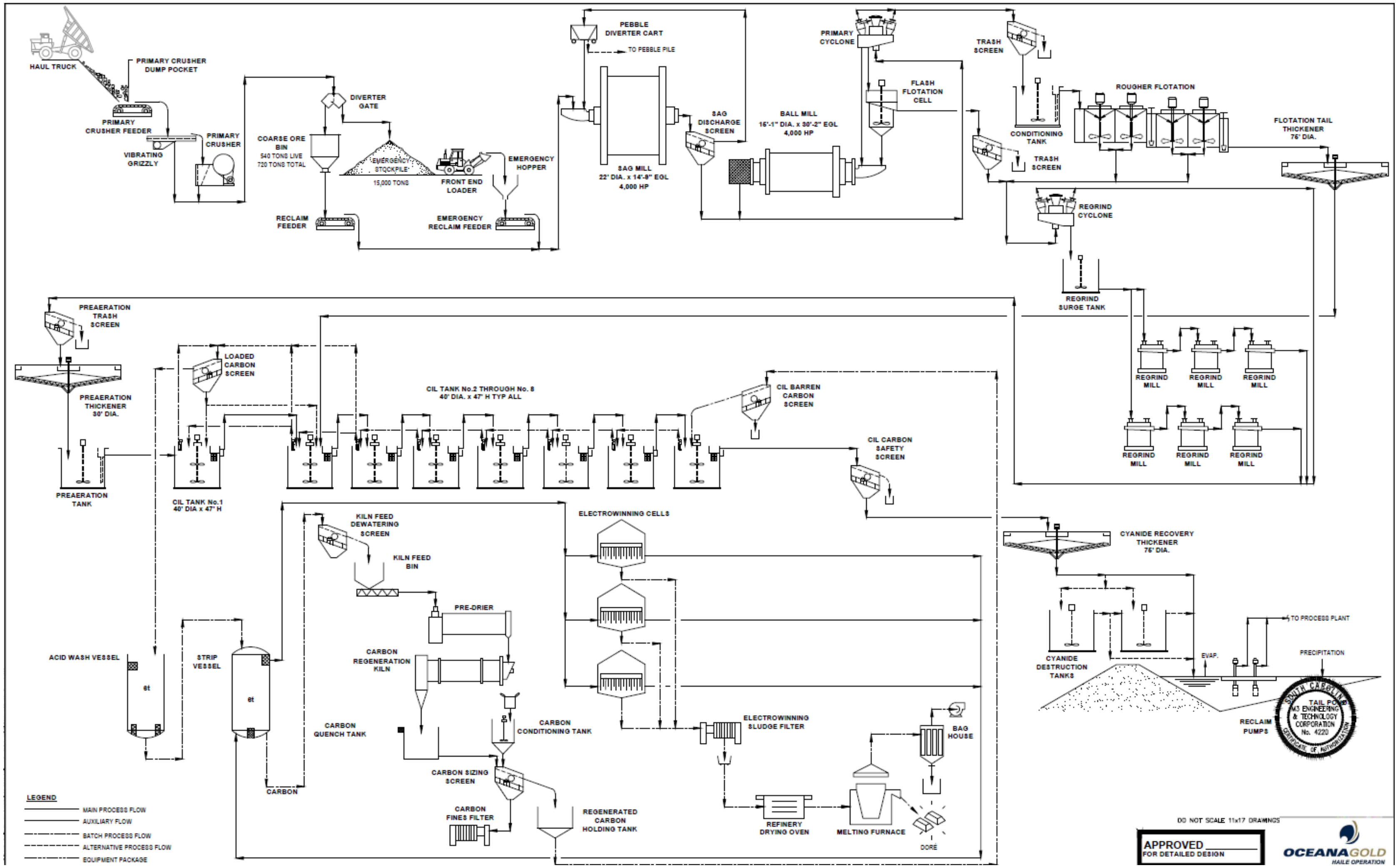
The detoxified tailing slurry is pumped for long term storage in a lined Tailings Storage Facility (TSF) and supernatant water in the pond is recycled for reuse back at the plant.

The plant has facilities for the storage, preparation, and distribution of reagents to be used in the process.

Reagents include flotation reagents i.e., potassium amyl xanthate (PAX), and MIBC as well as sodium cyanide, caustic soda, flocculant, copper sulfate, ammonium bisulfite, hydrochloric acid, lime and anti-scalants.

Small amounts of fresh and potable water make-up are required in the process but the main water requirements are satisfied by internal recycle from the thickeners and tailings decant water returned from the TSF.

The overall simplified process flow sheet for the current operation is shown in Figure 17-1.



Source: M3

Figure 17-1: Overall Process Flowsheet

17.1 Operation Results

The processing plant is currently operating in the commissioning and ramp up phase and as such there is insufficient data to validate the process model at the time of issue of this report.

However metallurgical recoveries are broadly in line with those expected and throughput has reached nameplate for the current circuit at times.

17.2 Processing Methods

In general, the processing methods will not change for the proposed expansion. A conventional flotation and cyanide leaching flow sheet will continue to be used at Haile. Additional equipment is added in some areas to meet the enlarged duty and some reconfiguration will be completed to cost effectively rectify weaknesses in the original Romarco design. Some equipment is proposed to be retained but repurposed and new equipment constructed in the original item location that is more suitable for the enlarged flow rate.

The key additions to the process plant for the expanded capacity required that were considered, compared to the existing process equipment, included:

- Secondary crushing and screening,
- Pebble crushing installation on existing SAG mill scats recycle,
- Tertiary (whole ore) grinding using a vertical ball mill,
- Extra flotation cell to maintain retention time,
- Primary regrinding stage using a vertical ball mill,
- New pre-aeration tank with oxygen injection,
- Expanded concentrate leaching,
- Additional acid wash and strip vessels, and
- Additional flotation tails and CIL tails thickening capacity.

The following section describes the modified plant operation.

17.2.1 Crushing and Grinding

RoM ore will continue to be fed from the existing tip bin using the installed apron feeder and will pass over the existing vibrating grizzly where fines will be scalped out and oversize material will feed the existing primary (jaw) crusher operating in open circuit exactly as per the current arrangement.

The primary crushed ore stream will be able to diverted by a modified chute from the head of the first extant crusher area conveyor onto a new conveyor and delivered via a scalping screen into a secondary crusher operating in open circuit. Screen undersize and secondary crushed ore will be combined and conveyed back to the existing coarse ore bin and stockpile arrangement.

A twin parallel screen and crusher arrangement was developed but it is anticipated that only a single line will be equipped initially and the second set of equipment installed later once an assessment of crushing equipment availability and productivity is able to be made.

Secondary crushed ore will similarly either be reclaimed from either the surge bin or stockpiled and re-fed by FEL into the reclaim bin and delivered to the upgraded grinding circuit.

The existing conveyors are suitably sized for the increased plant feed rate but some will require a speed increase to reduce loading per unit length to avoid spillage. A minor upgrade to the SAG mill feed conveyor drive is required to cope with the increased circuit capacity for new feed and the design pebble recycle rate.

The existing 2-stage grinding circuit that consists of a primary SAG and secondary ball mill will be supplemented with a pebble crusher and tertiary vertical tower mill to meet the increased throughout.

The SAG mill discharge grates design will be changed to include pebble ports and mill discharge is still fed to the existing vibrating horizontal screen. The screen is adequately sized for expansion and the screen oversize will be conveyed to a new pebble crushing circuit. A magnet is provided for tramp steel removal. The crushed pebbles will be delivered back onto the SAG mill feed conveyor.

Since plant start-up the lack of a pebble crushing capability has been identified as a key weakness in the design and plans have been initiated to fast track the installation of these specific modifications urgently during 2017 rather than delay until the plant expansion project is started.

The SAG discharge screen undersize is fed to the existing ball mill which will continue to operate in closed circuit, but with a modified hydrocyclone cluster. The overflow slurry from this cyclone cluster will feed a new tertiary grinding circuit

A portion of the existing ball mill hydrocyclone underflow will still be treated in a flash rougher flotation cell and the flash flotation concentrate pumped to a trash screen and flash flotation tails returning to the secondary ball mill.

The tertiary grinding circuit envisaged consists of a vertical ball mill (tower mill) operated in reverse closed circuit with its own hydrocyclone cluster.

The tertiary mill model was selected as conservatively sized with a generous installed power due to the uncertainty over the LOM variation in ore competency given the lack of detail in the mining schedule at the time. A smaller unit is potentially suitable and this is more likely if the design criteria for the circuit product particle size is altered based on operating experience gained in the near term.

The grinding circuit product slurry from the overflow of the tertiary tower mill's hydrocyclones will pass through a new trash screen and slurry sampler to feed the flotation circuit.

17.2.2 Flotation

The design flotation circuit residence time could be maintained with the installation of a new pre-rougher flotation cell located in place of the existing conditioning tank which would be removed.

However, since plant operations have started up at Haile, rougher flotation kinetics appear to be faster than expected and the extra flotation cell is not considered to be likely to be necessary at the time of compilation of this report.

The existing rougher flotation cell configuration can also be modified to allow operation of the last two cells in a scavenger configuration if required as part of the expansion though this is not a critical action.

The flotation tailings slurry will still be dewatered in the existing flotation tails thickener; an additional hi-rate thickener could be installed, operating in parallel, to meet the expanded plant capacity. However, the extant high-rate thickener capacity is limited primarily by the undersized feed launder and feed well arrangement and the obsolescent design of these components as opposed to the installed area.

It is expected that replacement of these components with a design specified for the flowrates expected for upgraded plant capacity using more appropriate modern technology will suffice and a complete new thickener may not be necessary.

17.2.3 Regrind

The existing pre-aeration thickener, which was installed to dewater reground concentrate slurry prior to leaching, will be repurposed to instead dewater the combined flash and rougher flotation concentrate slurry streams *prior* to regrind. The significantly coarser size distribution of the “as floated” concentrate is expected to display enhanced settling rates, improve overflow clarity and the extant thickener dimension is suitable for the increased design throughput.

Moreover, dewatering concentrate slurry prior to regrind allows more stable operating conditions for the regrind circuit to be created with a higher solids content in the feed which improves the overall site water balance.

The regrind circuit at Haile is an area that will benefit from a re-arrangement to bring the design more in line with proven operating practice.

The concentrate thickener underflow will be pumped to the existing but repurposed pre-aeration tank that will be used to provide surge currently missing between the flotation and regrind circuits.

The concentrate slurry in this surge tank will be pumped to a scalping hydrocyclone cluster and the coarser concentrate particles in the underflow stream ground in a new primary regrind tower mill operating in open circuit.

The primary regrind mill model selected is common to the tertiary (ore) grinding mill and is potentially as conservatively sized with a generous installed power due to the uncertainty over the concentrate regrind specific energy requirement

Primary regrind mill discharge slurry will be recombined with regrind cyclone cluster overflow in the existing small regrind surge tank and the combined flow pumped to the existing stirred mill detritors operated as currently configured (i.e., open circuit regrind mills in twin parallel trains of three units in series) but now in a secondary duty more suited to their design and media size, to produce the requisite concentrate regrind circuit product size.

Reground concentrate slurry will be pumped via a new trash screen to remove fugitive regrind media, to a new larger concentrate pre-aeration tank replacing the tank used for improved surge capacity described above. Oxygen will be added to the slurry using a closed circuit pumped flow high shear oxygenation device to enhance passivation of reactive metal sulfide mineral surfaces.

As well as these key changes, throughout this section of the plant general pump and piping upgrades would be required.

17.2.4 Cyanidation

Oxidized concentrate slurry could be pumped to a new carbon in leach (CIL) circuit with the residue from the concentrate CIL circuit pumped to the existing to the existing CIL tanks and combined with thickened flotation tailings slurry.

This was believed necessary to ensure sufficient residence time would be retained for the float tails leaching by converting the first in train single large concentrate CIL tank to be the first of eight flotation tails CIL stages.

This represents a high cost option and as such is non-preferred. Operational experience infers the concentrate leach kinetics are faster than predicted and preg-robbing is not anticipated to be of any significance.

In place of the proposed CIL a hybrid Concentrate leach arrangement is preferred obviating the requirement for carbon to be present, removes the cost for interstage screens, carbon transfer pumps etc.

As well as these key changes, general pump and piping upgrades would be required.

17.2.5 Carbon Handling

To manage the extra volume of loaded carbon produced from both the concentrate and flotation tailings CIL circuits in the expanded plant, the installation of an additional acid wash and elution column was allowed for. The current plant elution area structural steel was designed to readily accept an additional column for each application.

Since start-up, reasonable metal loading values were noted. Also, lower than anticipated plant feed silver grades compared to the geological resource model predictions and most importantly faster batch turnarounds in the elution circuit operation have been performed. This supports the deduction that the extra vessels would not necessarily be required. A second batch per day can be completed to make maximum use of the existing installed elution capacity then the second columns are redundant.

The extant electrowinning cells were sized based on the flowrate through the strip so running a second strip per day (one at a time), will not require any additional cells to operate.

The existing regeneration kiln was sized to handle a maximum throughput that is believed adequate for the plant expansion.

17.2.6 Tailings

As there will be an increased volume of CIL tailings slurry, some will be dewatered in the existing cyanide recovery thickener; but an additional new thickener will also be installed, operating in parallel, to meet the expanded plant capacity. Thickener overflow will be recycled via the plant reclaim water system to the first flotation tails CIL tank to provide dilution for slurry density control and make use of residual cyanide values in solution.

Thickener underflow slurry will be pumped to the existing cyanide destruction circuit. The removal of residual cyanide in the leach tail stream will be achieved using the two existing Detox tanks operating in parallel. The installed detox tank volume will provide adequate residence time for the increased flowrate.

The existing detox compressor will be able to supply sufficient air to feed the detox circuit. If needed, the detox air system can also be supplemented by connecting via a pressure reducing valve to the leach compressor main. If oxygen is used in the new concentrate pre-aeration tank in place of compressed air, spare capacity can be used in the detox system.

Tailings pumps and piping will be upgraded to deliver the expanded tailings slurry flowrate to the existing TSF as the dam wall is raised to increase storage volume. This requirement is included in the LOM plan sustaining capital allowances.

Water from the tailings pond will be recycled for reuse in the process using the existing decant pumping installation.

17.2.7 Water

Both concentrate and tailings thickener overflow solutions will be recycled back to the grinding and flotation circuits via the internal reclaim water tank.

The original concept of using the TSF decant water as wash in the CIL tailings thickener is flawed and will be removed from the design.

Gland seal water pumping requirements can be met with existing pumps. Some gland water pumps may need to be repurposed or upgraded.

As well as these key changes, general pump and piping upgrades would be required.

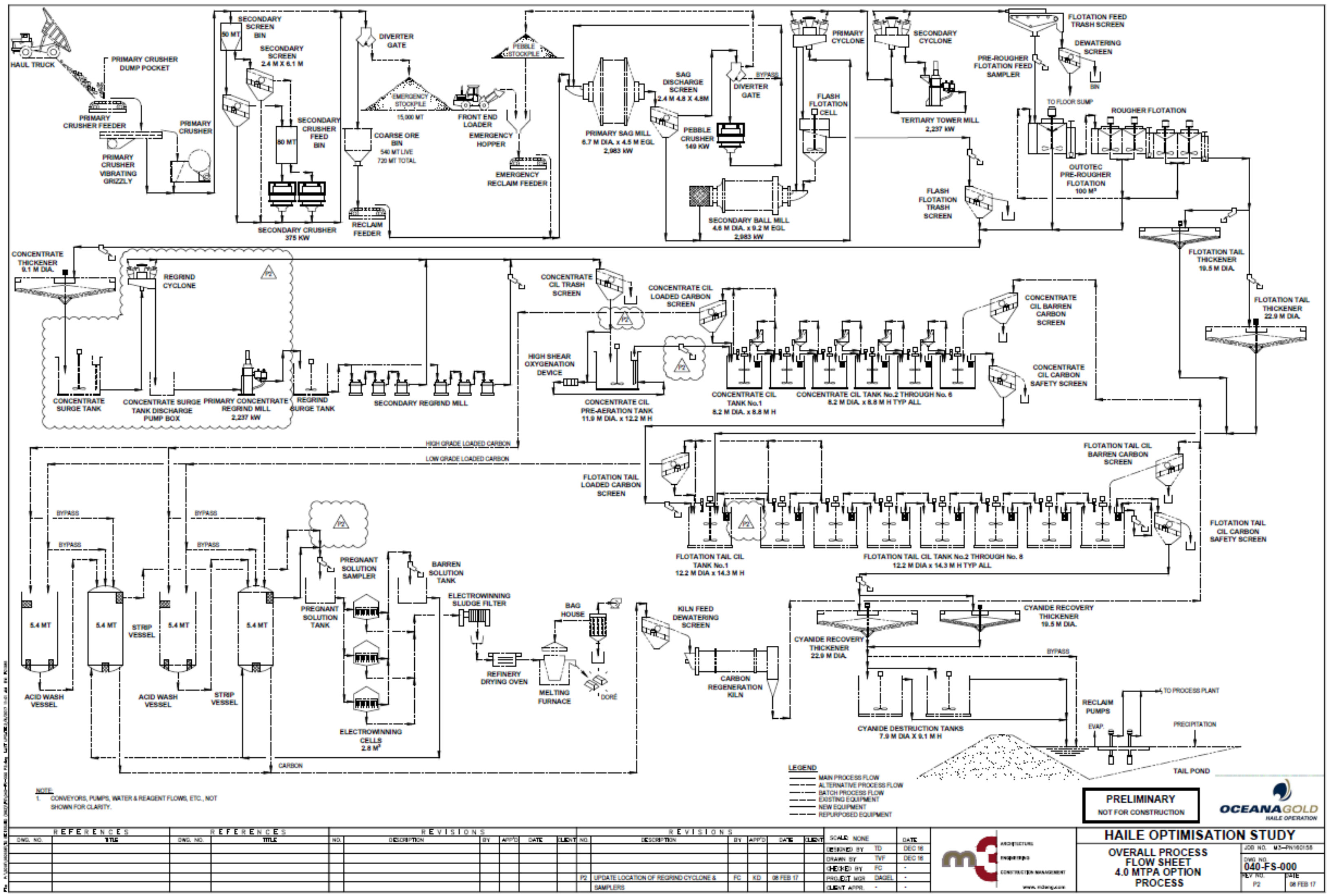
17.2.8 Reagents

The storage, preparation, and distribution of reagents to be used in the process will utilize existing equipment for flotation collectors, frother, sodium cyanide, caustic soda, copper sulfate, ammonium bisulfite, hydrochloric acid, lime, antiscalant, etc.

A new flocculant solution make-up system will be installed for the additional CIL tail thickener.

17.3 Flowsheet

The overall simplified process flow sheet for the proposed expanded plant is shown in Figure 17-2.



Source: M3

Figure 17-2: Plant Expansion Overall Simplified Process Flowsheet

17.4 Consumable Requirements

The consumable requirements for the expanded process plant are of the same type as the current operating plant uses as the recovery process will not materially change.

The volumes of certain consumables will increase proportionally. Also, different grades, qualities or specification of grinding media will be required depending on the exact details of the regrind mills selected for the upgraded circuit.

Reagent consumption rates and grinding media consumption rates for full scale plant operation were estimated from the results of the RDi test work are shown in Table 17-1 and Table 17-2.

Table 17-1: Process Reagents

Item	Rate kg/t ore
Collector, Potassium Amyl Xanthate	0.07
AERO 404 (or equivalent)	0.07
Frother, Methyl Isobutyl Carbinol	0.05
pH Modifier, Lime	2.07
Sodium Cyanide	1.07
Flocculant	0.06
Antiscalant	0.01
Hydrochloric Acid	0.10
Lead Nitrate	0.006
Copper Sulfate	0.02
Ammonium Bisulfite	0.46
Carbon	0.034

Source: OceanaGold

Table 17-2: Grinding Media

Item	Rate Kg/t whole ore
Grinding Balls, SAG Mill	0.50
Grinding Balls, Ball Mills	0.50
Grinding Media (Steel), Primary Regrind Mill	0.19
Grinding Media (Ceramic), Secondary Regrind Mills	0.15

Source: OceanaGold

18 Project Infrastructure

18.1 Tailing Storage Facility

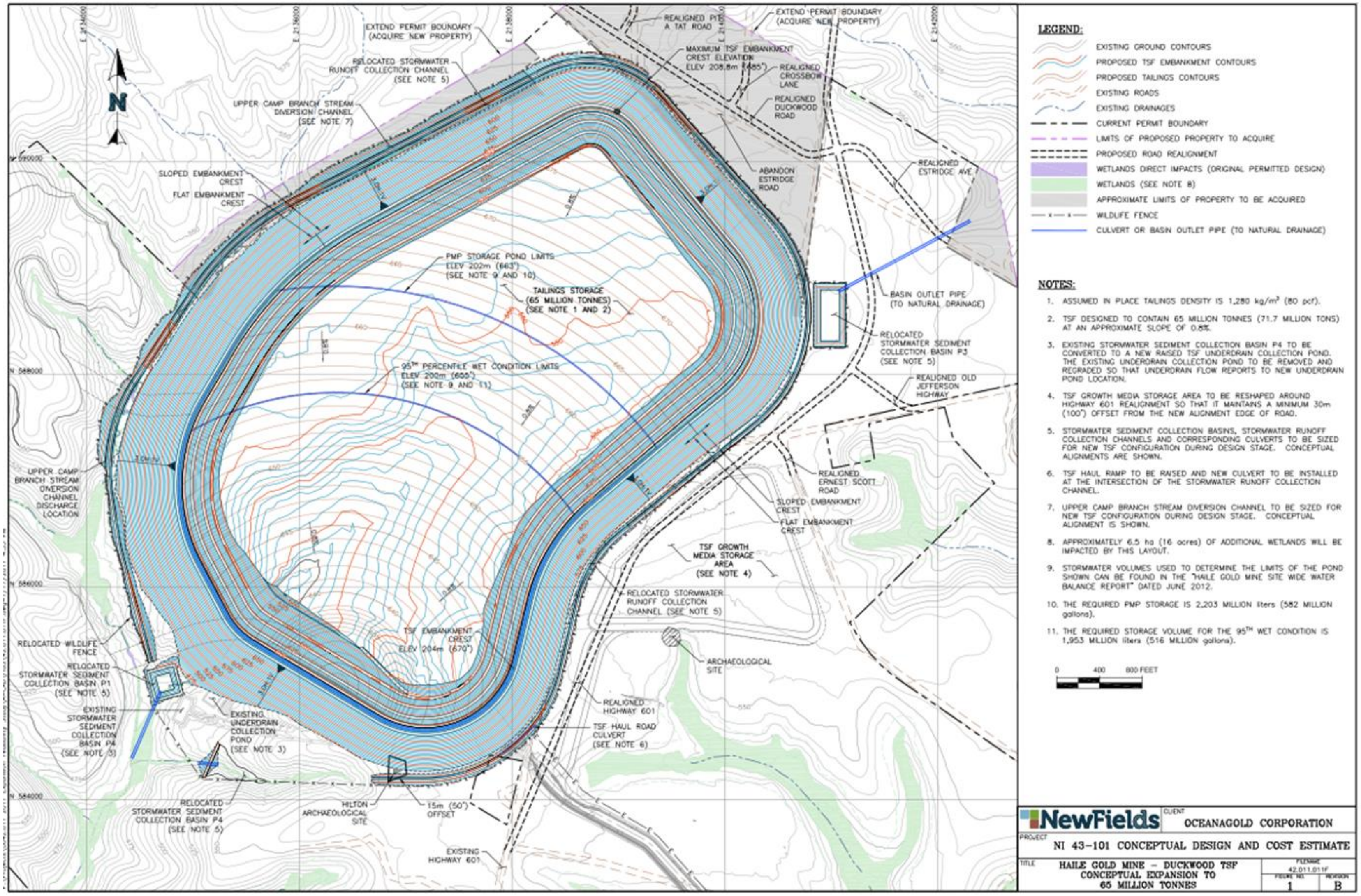
To meet the revised tailings storage capacity requirement, NewFields has evaluated the permitted ultimate Duckwood TSF design and has determined that the embankment crest elevation needs to be raised from the current permitted height of 192 m to 204 m. As a result, the storage capacity of Duckwood TSF will increase from 36 million tonnes to 65 million tonnes. During this evaluation, NewFields sloped the embankment crest in the north corner of the facility, which is outside the predicted limits of the maximum storm water pond, to optimize tailings storage capacity. All other design concepts; which include basin lining, embankment section and materials, tailings deposition scheme and freeboard requirements, will remain the same as the permitted facility. The revised Duckwood TSF layout with ultimate footprint is presented in Figure 18-1. The construction of the Duckwood TSF will be phased in five stages (downstream embankment design) as presented in Figure 18-2 and includes Stage 1 which is already constructed.

In addition to raising the embankment, the footprint of the ultimate facility will require the following improvements that will be constructed during Stage 4:

- Realign portions of the following local roads and highways:
  - US Highway 601
  - Duckwood Road
  - Old Jefferson Highway
  - Estridge Avenue
  - Pit a Tat Road
  - Crossbow Lan
- New haul truck overpass over the realigned Highway 601
- Reconstruct all perimeter runoff collection channels
- Relocate and construct a new channel for upper Camp Branch creek.
- Remove existing stormwater sediment collection basin P2 and the current TSF underdrain collection pond
- Reconstruct perimeter stormwater sediment collection basins: P1, P3 and P4
- Repurposing the existing P4 stormwater sediment collection basin into the new TSF underdrain collection pond
- Reshape the Existing TSF Growth Media Stockpile
- Relocate the Hilton Archaeological Site

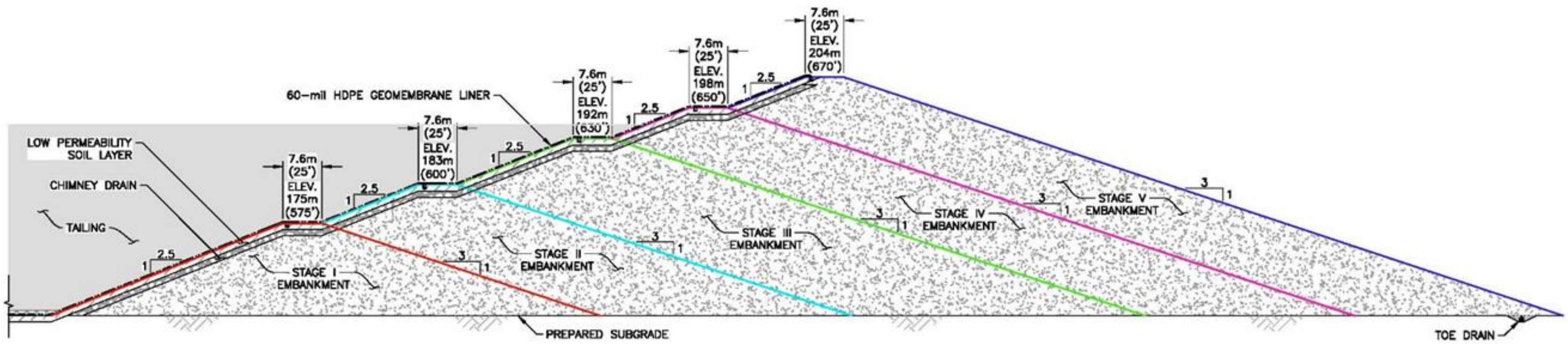
The deposition of tailings into the TSF will be via a HDPE pipeline located around the perimeter of embankment crest. Deposition will occur from multiple spigots inserted along the tailings distribution line. The deposition locations will be moved progressively along the distribution line, as required, to maintain slightly graded deposition of tailings towards the decant pond that is located in the southeast corner of the facility. Water from the decant pond will be recycled back to the mill for make-up water and will be reclaimed by utilizing either vertical turbine pumps mounted on a floating barge above the decant pond or skid mounted pumps to be located on the ramp within the southern end of the decant pond.

The TSF is designed as a zero discharge facility. In addition to the anticipated tailing storage and operating pool requirements, the facility is designed to contain the Probable Maximum Precipitation (PMP) storm event and an additional four feet of freeboard at all times.



Source: NewFields

Figure 18-1: Tailing Storage Facility Layout



Source: NewFields

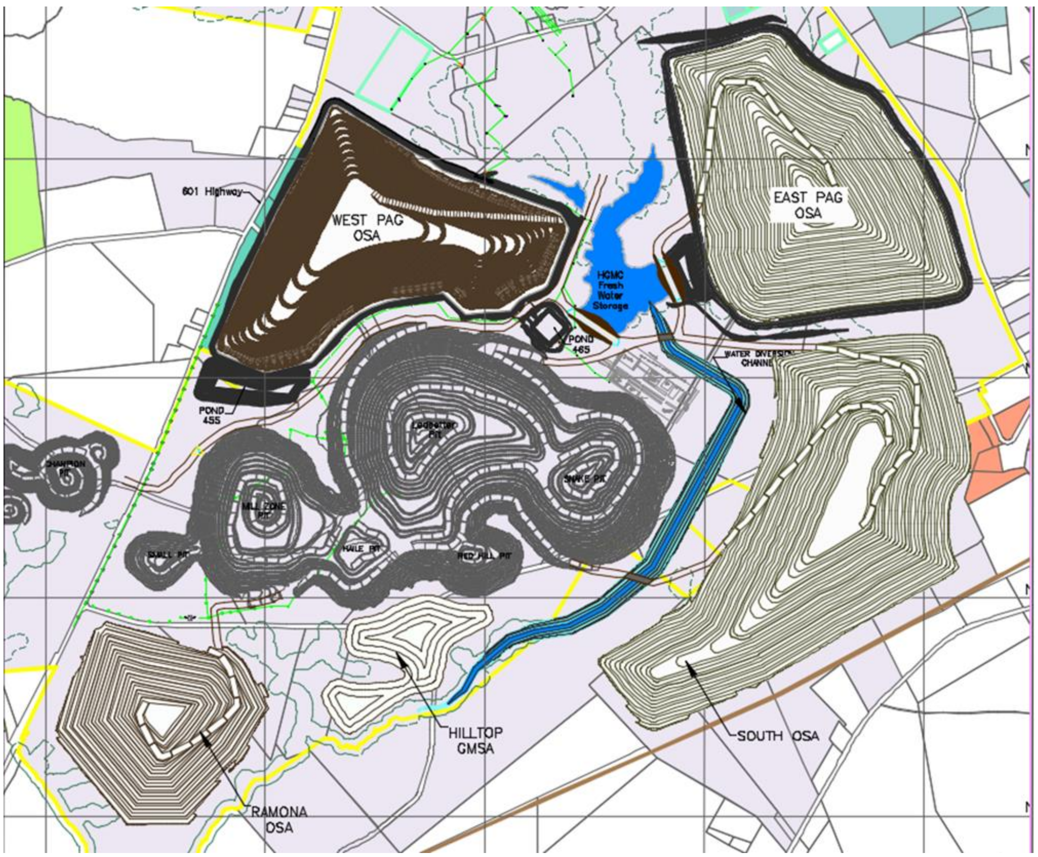
Figure 18-2: TSF Typical Section showing Downstream Method of Embankment Construction

18.2 Overburden Storage

During the mine life, four different Overburden Storage Areas (OSAs) will be utilized for the storage of approximately 481 million tonnes of material generated from the pit development. The material generated from the development of the pits will be classified as either potentially acid generating (PAG) or non-acid generating overburden material. The PAG material will be stored exclusively within the West PAG and East PAG OSAs. The other two OSAs (Ramona and South OSA) are designated for storing non-PAG material. The OSAs will be developed according to the pit progression and the final footprint of the OSAs is presented in Figure 18-3.

Prior to construction of the OSA's, the footprints will be timbered. Grass lined sediment collection control channels will be constructed around the footprint of each OSA. Sediment control structures will be constructed at the outfall of the sediment control channels for each facility. Water retained within the ponds is routed through a low-level riser pipe to an adjacent drainage for collection and reclaim.

All of the OSAs will be developed with a final reclaimed overall 3(H):1(V) slope.



Source: OceanaGold

Figure 18-3: Overburden Storage Areas Plan

18.2.1 Potential Acid Generating (PAG) Overburden Storage Areas

The expansion of the open pit is expected to increase the amount of PAG material that is required to be stored in a fully geomembrane lined facility. Under the current mining permit, Johnny's PAG (JPAG)

OSA is designated as the dedicated facility for storing PAG material with a maximum capacity of 41.7 million tonnes. It is estimated that the revised mine plan will generate 221 million tonnes of PAG material which will require two separate PAG OSAs.

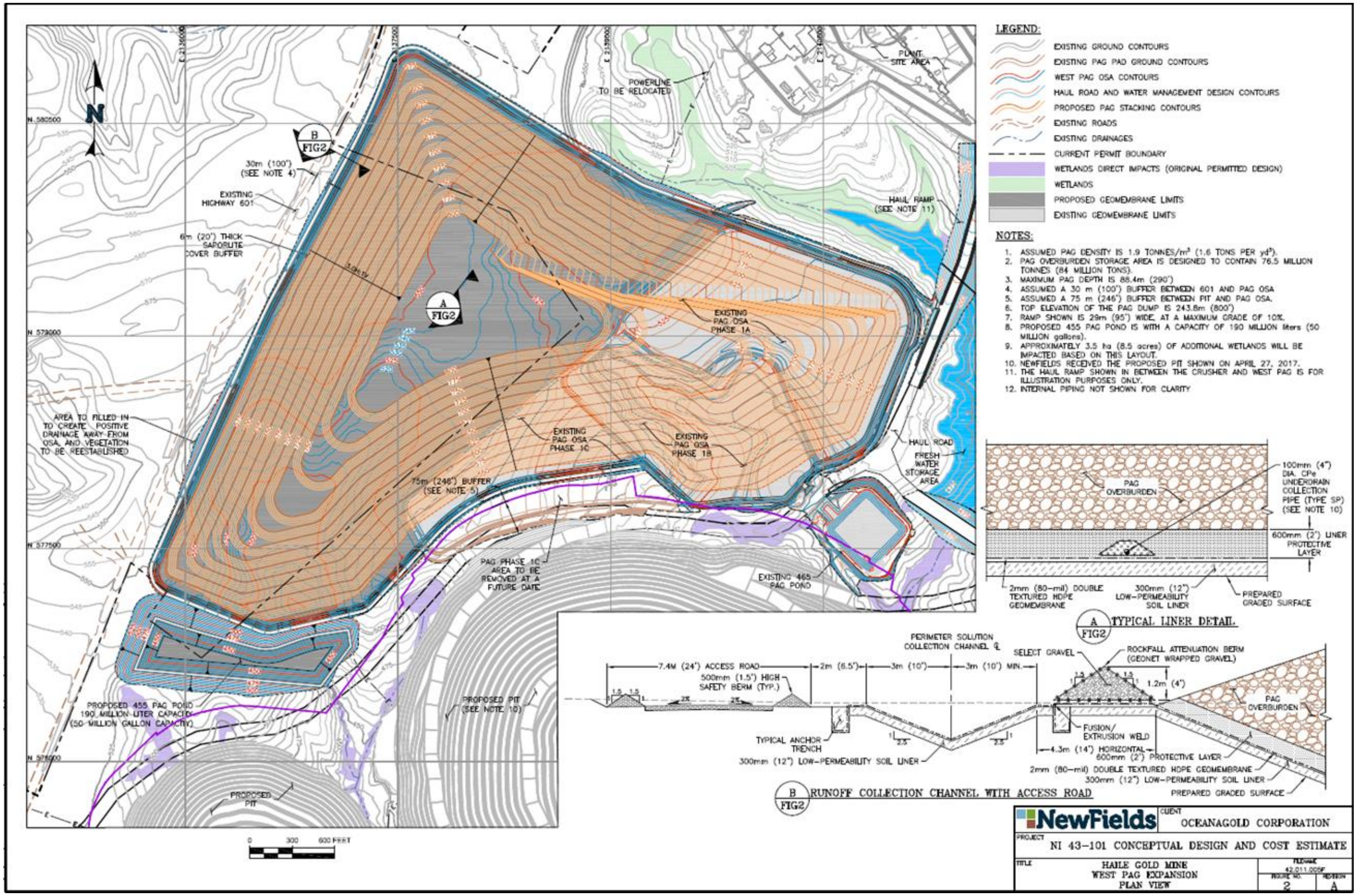
OceanaGold engaged NewFields to evaluate expanding the existing JPAG OSA to determine the maximum volume of PAG material that can practicably be stored. The expanded facility (Figure 18-4) is named “West PAG OSA” and has a total capacity of 76.5 million tonnes. Similar to the permitted JPAG facility, the expanded base pad of West PAG will be lined with a composite lining system utilizing a low permeability soil layer overlain by a geomembrane. The geomembrane will be covered with a 600-mm drainage layer. A pipe network will be installed within the drainage layer to collect and transmit infiltration through the PAG material and direct it into the contact water collection pond.

The West PAG OSA will require a second contact water collection pond, referred to as 455 Pond, which will replace the permitted 469 Pond. The interior of the proposed pad expansion will drain to 455 Pond whereas the perimeter runoff collection channel will drain to either the 465 (existing) or 455 Pond. In accordance with the current water permit, the PAG solution and storm water collected in the 455 Pond will be pumped to the 19 Ponds for treatment and release, or for use in the milling process. The 455 pond is sized to hold 190 million liters which equates to the predicted runoff from the 100-yr/24-hr storm event.

The ultimate footprint of West PAG OSA will have an overall footprint of approximately 126 hectares and the PAG material will be loaded with an overall slope of 3(H):1(V).

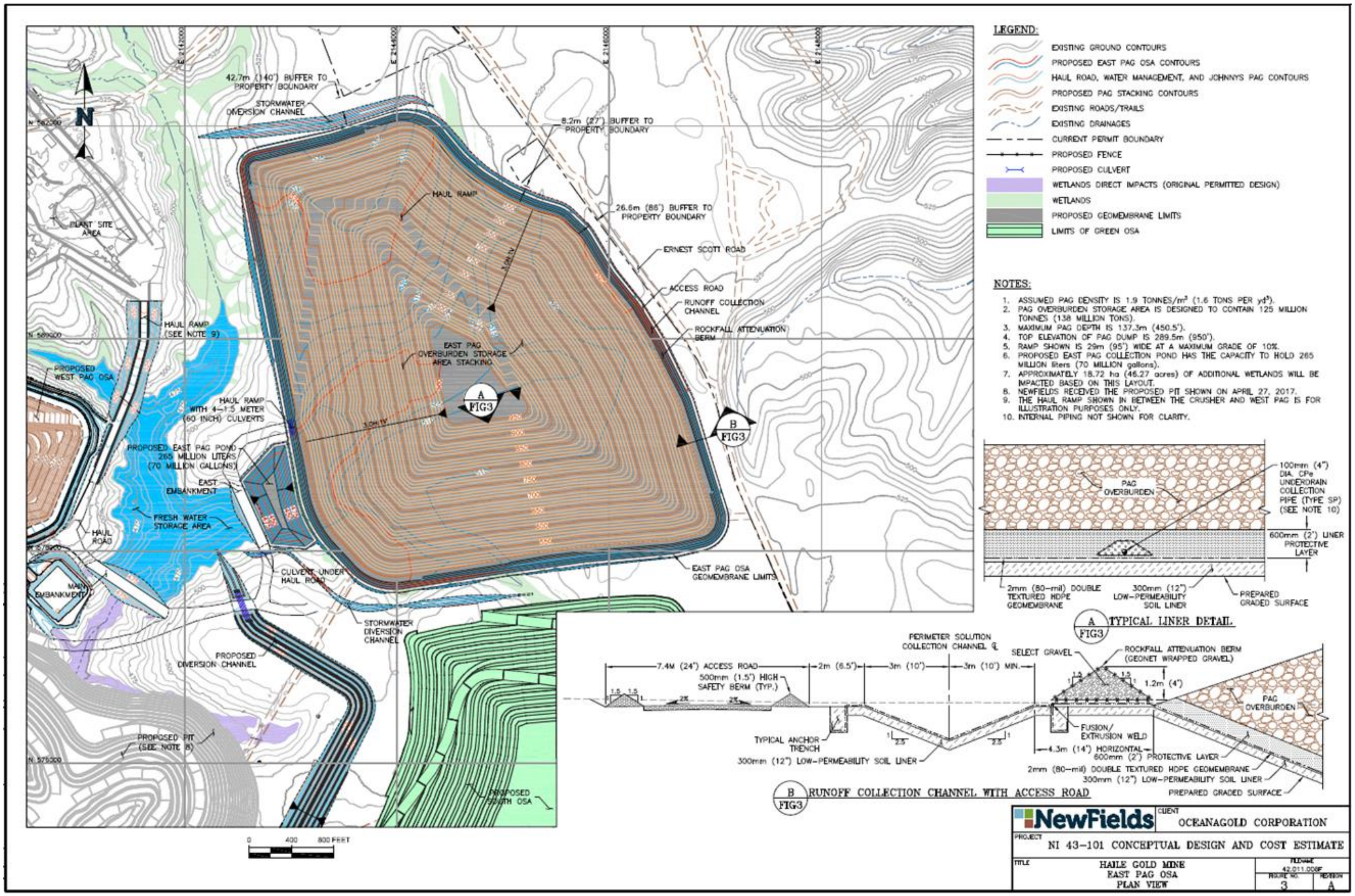
The second PAG OSA (named as East PAG OSA and shown on Figure 18-5) is located in the East of the Haile site and has an overall footprint of 141 hectares and a total capacity of 125 million tonnes of PAG material. Similar to West PAG OSA, the East PAG OSA will be constructed and loaded in accordance with the existing JPAG facility. The Contact Water Collection Pond for East PAG OSA is sized to hold 265 million liters of water which equates to the predicted runoff from the 100-yr/24-hr storm event. The PAG solution and storm water collected in the East PAG Pond will be pumped to the 19 Ponds for treatment and release, or use in the milling process.

Prior to construction of West and East PAG OSAs, the footprint will be stripped of vegetation and topsoil. Topsoil materials will be stored in growth media area for future use in accordance with the HGM reclamation plan.



Source: NewFields

Figure 18-4: West PAG Overburden Storage Area



Source: NewFields

Figure 18-5: East PAG Overburden Storage Area

18.3 Site Wide Water Management

A site wide water management plan is in place for the existing operation. Water management facilities covered in the plan include retention structures, diversion channels, culverts, conveyance pipes, sediment collection channels and sediment control basins. These facilities provide for erosion protection, sediment control for site-wide surface water runoff arising from stormwater and diversion of existing streams around and beneath site infrastructure (e.g. roads, stockpiles, plant site and mine pits. Sediment ponds are used to reduce sediment loads and Water Treatment Plant is used to treat water collected from contact areas before releasing water back into natural drainages. Year on year water management will take into consideration the dynamic design life of the mine by evaluating each water management structure at the most critical design phase with the greatest peak discharge.

The existing site wide water management plan will be updated to reflect changes to the mine plan. With the revised mine plan, the incoming water from Upper HGMC will be detained to create a Fresh Water Storage Area (FWSA) at the upper reaches of the watershed as shown in Figure 18-6. As shown in Figure 18-7, the FWSA will be made up of two embankments, the Main Embankment located upstream of the proposed pit and the East Embankment located west of the proposed East PAG OSA. Both Main and East embankments will be geomembrane lined over a low-permeability layer and a chimney drain. Along the upstream embankment toe, a low permeability soil cutoff trench will be constructed. The embankments for the FWSA will be jurisdictional water retaining dams and will require the appropriate state permits. The FWSA has capacity to store approximately 590 million liters of fresh water

To maintain the minimum flow of 15.9 liters per second in HGMC, per the existing mining permit, a siphon off the mill return pumping system will be designed and installed near the Main Embankment should the water level drop below the diversion channel inlet elevation during dry spells. The overflow from the FWSA will be diverted around the pits in a diversion channel.

The diversion channel is designed to pass the storm water flow produced by the ½ PMP event. Where the diversion channel’s alignment parallels the proposed ultimate pit shell, the cross-section includes a geomembrane liner to minimize the potential of recharging groundwater near the pit. Everywhere else, the cross-section of the diversion channel is lined with a reinforced turf mat to the limits of the flow depth resulting from the ½ PMP event. The diversion channel outlets into a grouted riprap apron in an unnamed natural ravine feeding into HGMC downstream of the proposed pits and upstream of the confluence with Little Lynches River.

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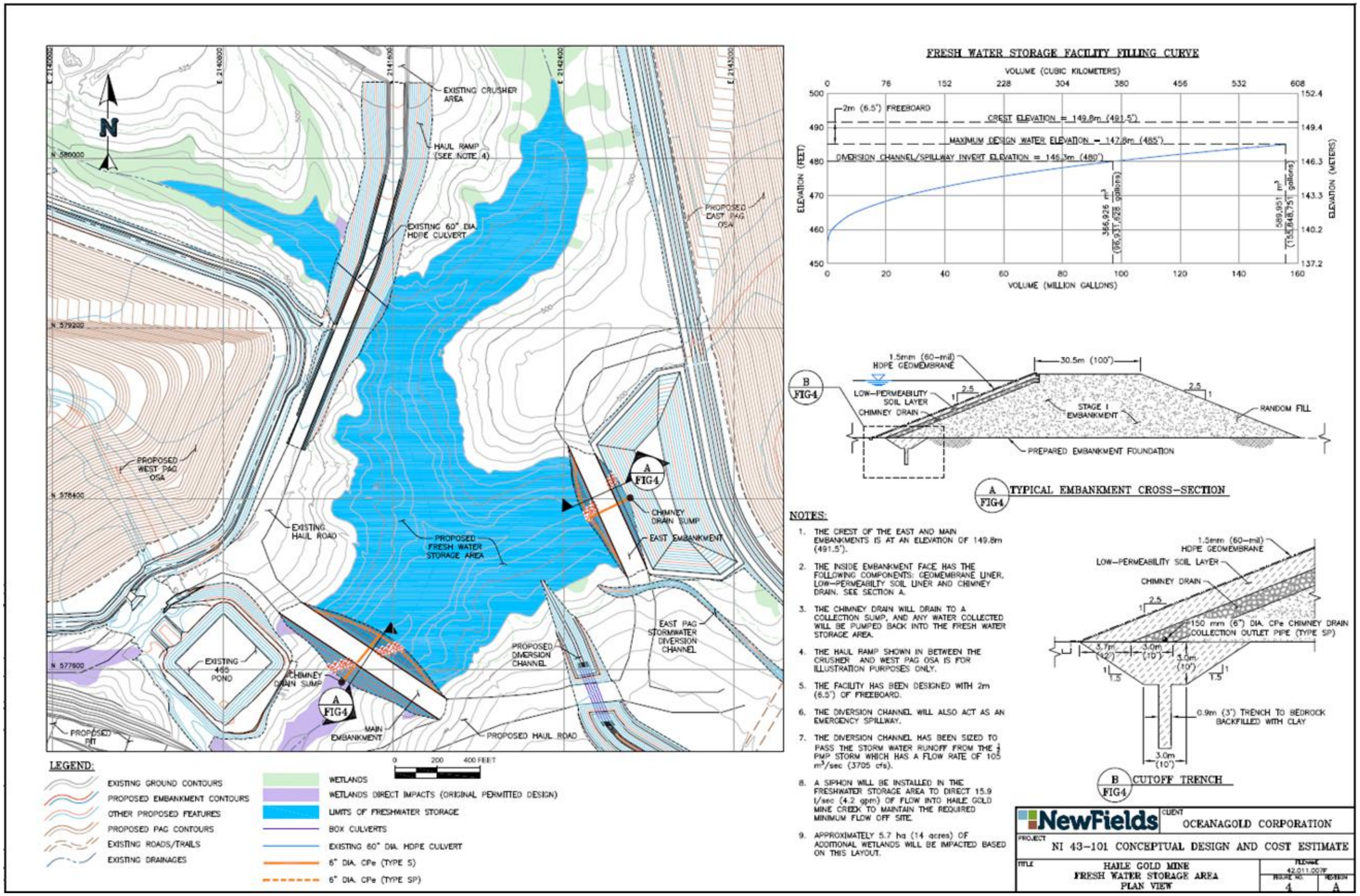
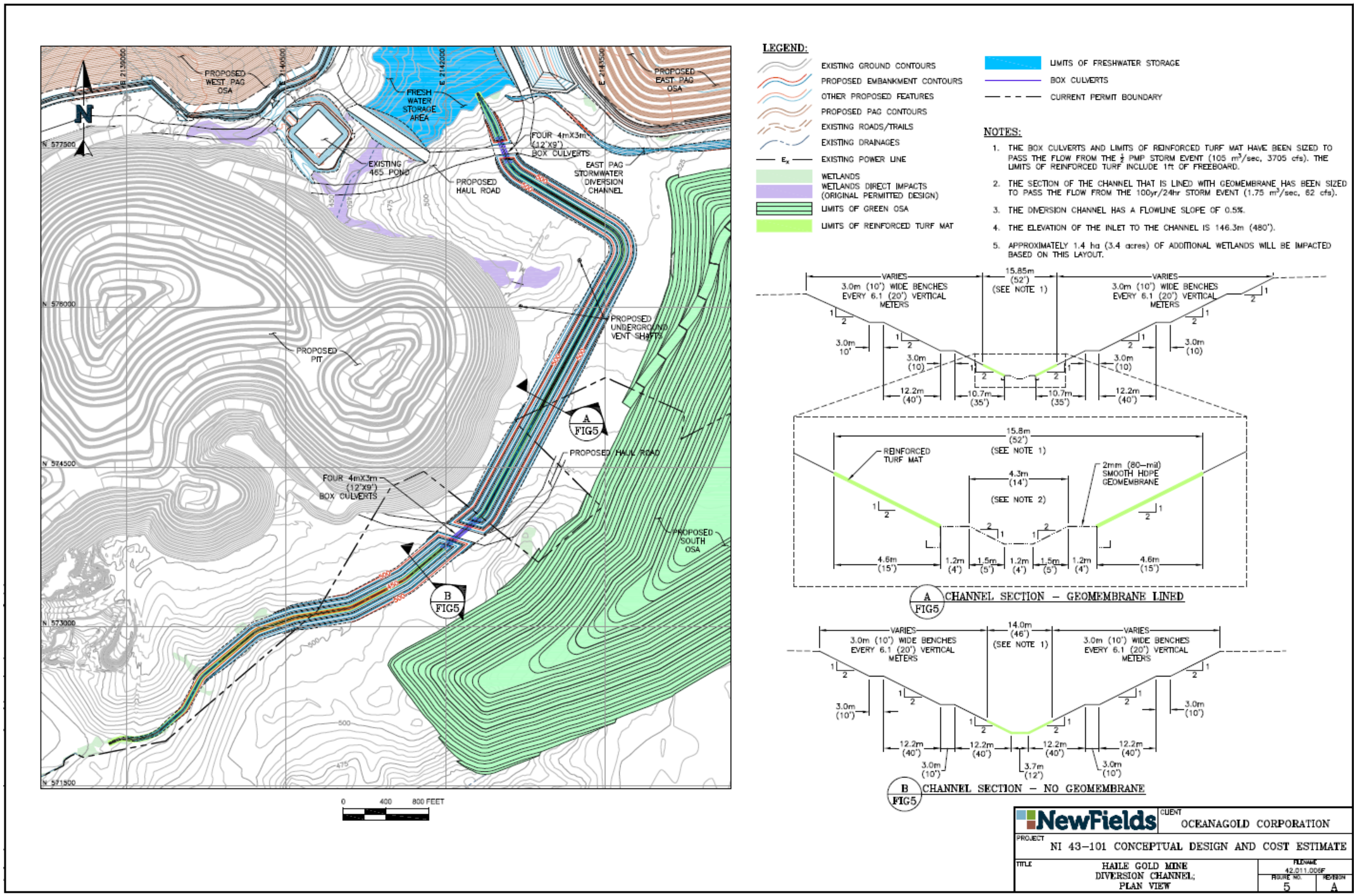


Source: NewFields

Figure 18-6: HGMC Fresh Water Storage Area

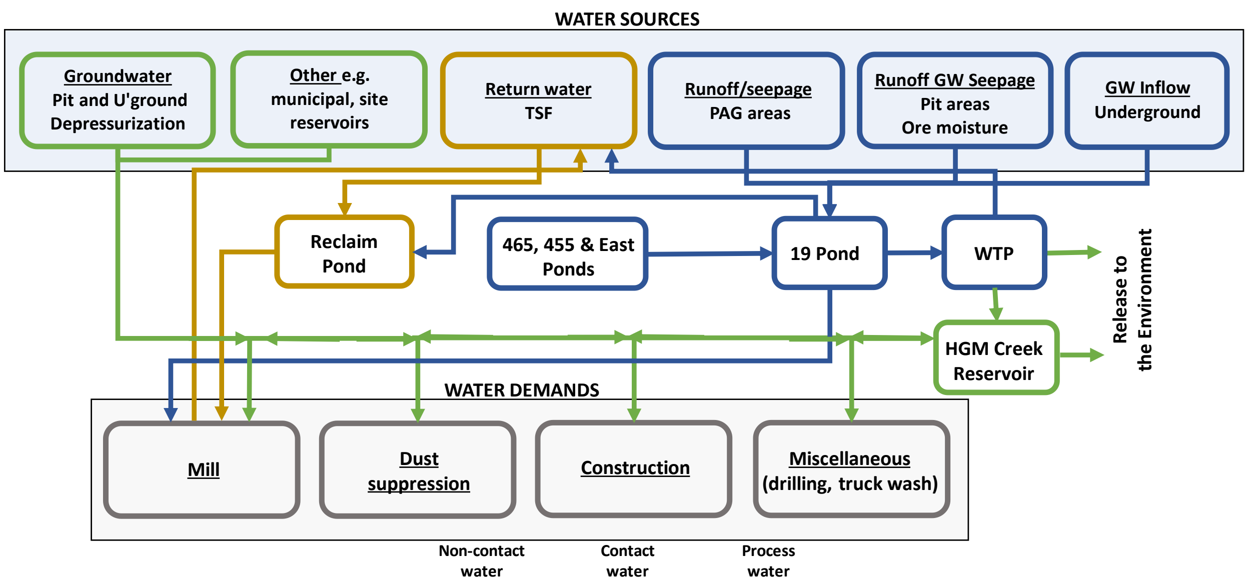


Source: NewFields

Figure 18-7: Water Diversion Channel Layout

18.4 Site Wide Water Balance

A site wide water balance was developed for the proposed Haile operations as a water management tool for the mine, particularly to aid in the planning, design and operation of the Mill, TSF and PAG overburden storage areas, as well as water storages and the control and treatment of contact and non-contact mine water. Probabilistic analyses using GoldSim software have been undertaken to consider possible implications for all major facilities expected to add water, store water, use water and treat water in response to rainfall variability (extreme rainfall events and extended dry periods), mine water production rates (as they change in relation to mine pit and underground mining developments) and mill throughput. Results from the study provide a variety of potential outcomes allowing risk-based decision making regarding water management. Figure 18-8 presents a schematic showing the site water interactions.



Source: CDM Smith

Figure 18-8: Site Wide Water Balance Model Schematic

As outlined in Figure 18-8, there are three source water types available on site – process water, contact water and non-contact water:

Process water can come from:

- Free water in the TSF;
- Underdrain from the TSF;
- Any water in the Mill process stream; and
- Natural moisture in the processed ore after it enters the process circuit.
- Make-up water sourced from contact or non-contact water generated on site.

Contact water can come from:

- Runoff and underdrain from PAG OSA and Low Grade Ore Stockpile;
- Direct precipitation and runoff accumulating in the active and inactive pits; and

- Crusher pad and coarse ore stockpile containment areas.
- In-pit and underground mine sumps.

Contact water can be used in the process as make up water, or be treated in the Water Treatment Plant prior to discharge to the environment.

Non-contact water, sourced from site areas and facilities that haven't been in contact with potentially contaminating activities, will not require treatment prior to discharge to the environment but can also be used a process make up water. Sources of non-contact water include:

- Groundwater from pit depressurization;
- Surface water from Ledbetter Reservoir;
- Municipal water;
- Runoff from Topsoil Stockpiles;
- Runoff from Overburden Storage Areas;
- Groundwater from underground depressurization;
- Runoff from Undisturbed Ground;
- Run-on from Up gradient Areas;
- Runoff from TSF Outer Perimeter; and
- Runoff from the Plant Site (process water is contained within the process).

The results of the site wide water balance analysis indicate that under normal and moderately extreme conditions, there will be adequate water returns from the TSF, surface water storages and mine depressurization wells to meet mine and process water demand.

Following cessation of mining activities, a water body will form in the pit void(s) as a result of groundwater inflow and incident rainfall. The level to which a pit lake will recover will be controlled by evaporation and the surface area of the recovered pit lake(s). A new dynamic equilibrium will be reached when evaporation losses are offset by water inflows. Preliminary modeling predicts the pit lake surface will recover to a quasi-steady state after many decades to an elevation around 20 metres below the pit rim and around 15 m below the pre-mine water table. Approaches to developing and appropriate pit closure strategy are now being considered.

18.5 Surface Roads and Bridges

A new concrete bridge type overpass will be constructed over the realigned Highway 601. The primary purpose of the overpass is to facilitate the haulage of construction fill material from the Mine to the TSF. The bridge will be designed for fully loaded haul trucks. In addition, the bridge will be used to carry the tailings delivery line across Highway 601 from the Process Plant.

18.6 Underground Access

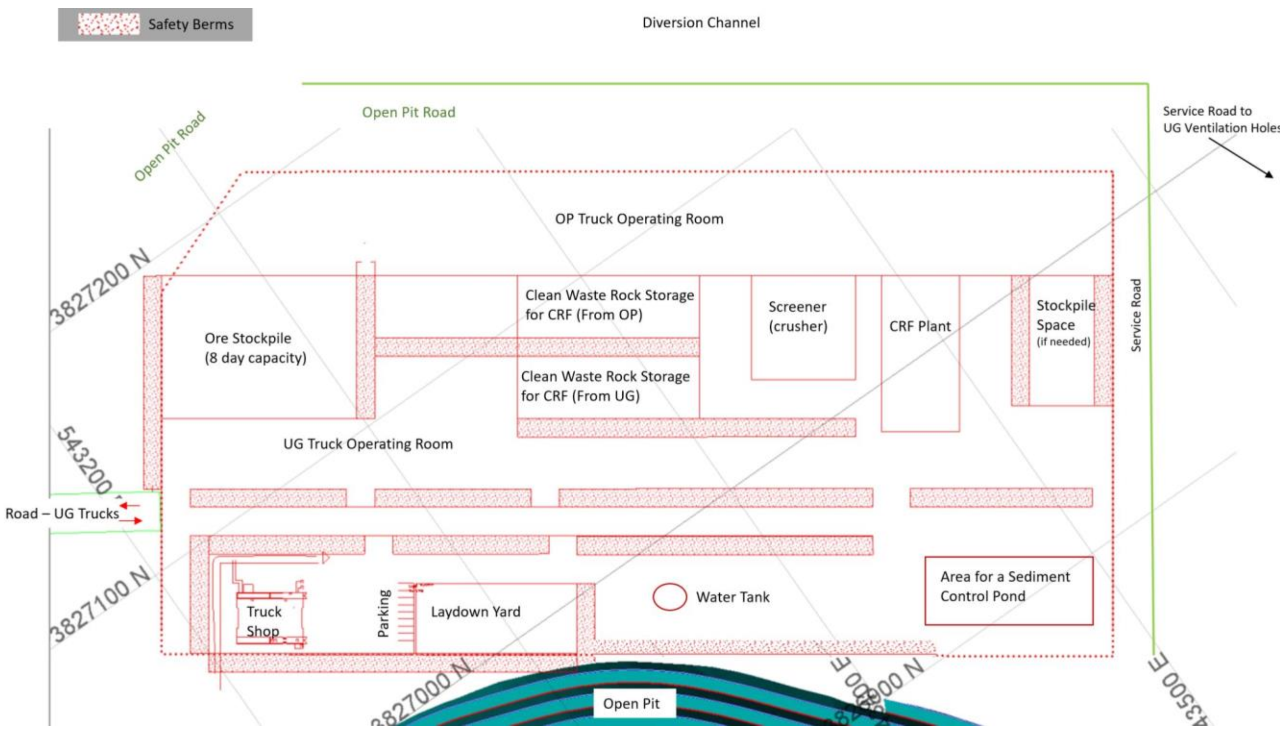
The underground will be accessed through the highwall in the Snake pit. The haul road accessing the portal will be from an open pit ramp that will tie into the existing haul road from the plant to the Snake pit near the existing Pond 465. The haul road will provide access to the underground yard for ore and waste movement from the underground as well as a haulage road for CRF back to the underground. The haul road and bench from the open pit will create enough width to allow the installation of required fans and electrical portable substations during the construction of the portal and decline as well as ventilation fans. The fans and substation will be removed once permanent facilities at the vent raise are established.

18.7 Ancillary Facilities

The project will utilize existing facilities such as maintenance workshop, truck shop and offices to support open pit and process plant operation. However, the inclusion of the underground operation will require additional facilities as follows.

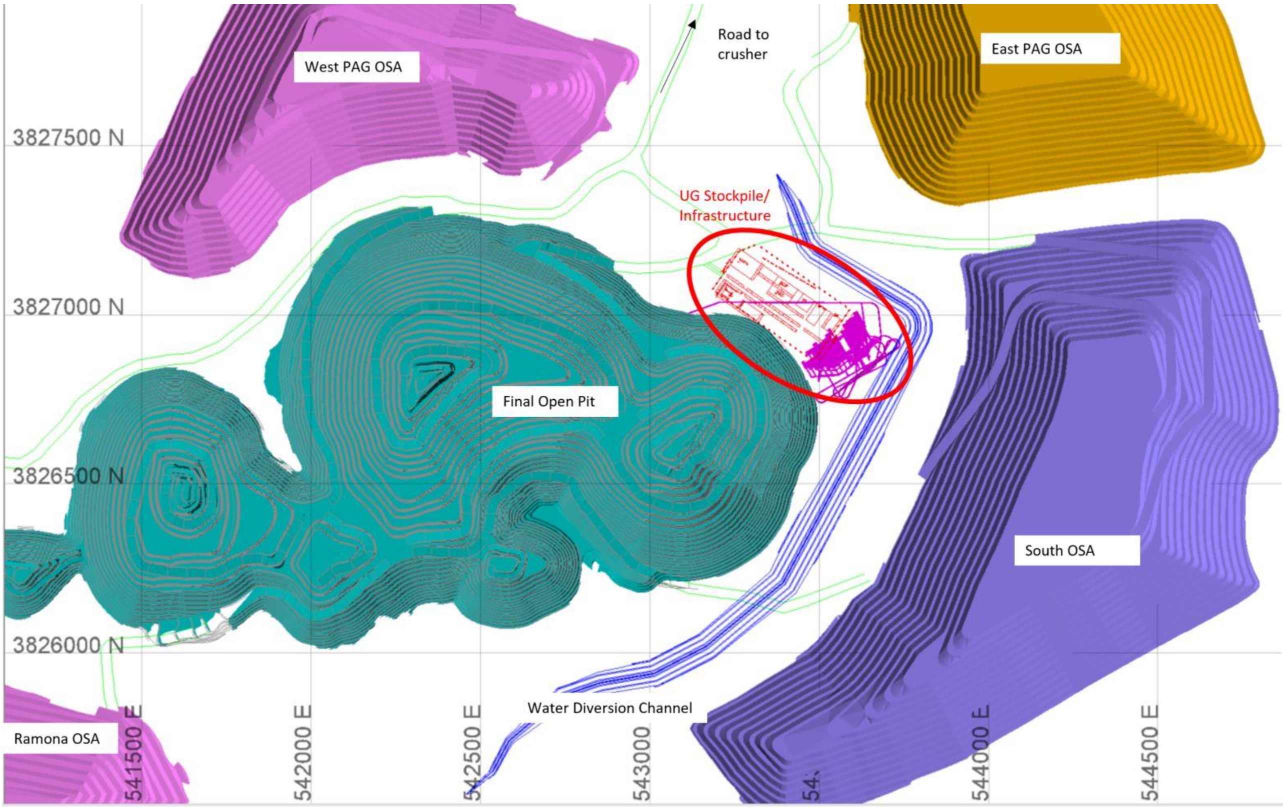
18.7.1 Underground Support Facilities:

The underground operation will be supported through a yard that will contain the underground ore stockpile, the underground waste stockpile, crushing/screening plant, CRF plant, and an underground maintenance shop as shown in Figure 18-9. The location of the UG yard is shown in Figure 18-10.



Source: SRK

Figure 18-9: Underground Surface Infrastructure Detail



Source: SRK/OceanaGold

Figure 18-10: Underground Surface Infrastructure Location

The underground shop will be a repurposed shop that is currently being utilized for the open pit fleet. It will include containerized storage and a cover that will allow two to four pieces of underground equipment to be worked on.

The yard will provide a location for stockpiling the underground ore that will be re-handled into surface ore trucks and hauled to the plant for processing. The yard will also provide a location for stockpiling waste material from the underground that will be either processed into CRF or hauled to the appropriate waste storage area. A small parking lot to support the shop will also be available. The yard will also provide a laydown area during the construction process for the underground mine and will house contractor trailers during construction.

A dry/change house and underground office building will be constructed near the existing office trailers area to support the underground operation. The location of these facilities is shown in Figure 18-11. The facility will house approximately 65 persons on a typical day shift. The back shifts and weekends would typically have about 47 people per shift. Employees will be transferred to the underground mine yard by van or bus.

Source: SRK/OceanaGold

Figure 18-11: Mine Offices/Dry Location (1,000m grid lines)

18.8 Power Supply

The total power demand for the site (including Horseshoe underground operation) is estimated to be 28.5 MVA. The study undertaken by Lynches River Electric Cooperative confirmed the availability of power to site with some minor upgrades to the existing 69 kV substation and transmission line.

For the underground operation, the existing power line from the main substation to Pond 465 will be extended to the underground yard and a transformer will serve the CRF plant. An additional transformer will provide power to the underground shop and a transformer will provide power to the portal area. The power line from the underground yard area will be extended to the vent raise area to provide primary mine power once the ventilation raise is complete. A transformer will provide 13.8 kV power to service the mine. The mine power system is described in section 16.2.11.

18.9 Water Supply

Fresh water required for dust suppression and the process plant will be supplied by the pit dewatering wells. The excess water from the dewatering wells will be pumped to the FWSA before releasing to the environment. Mill make up water will be supplied by the return water from the tailings facility. Surface runoff into the pits and PAG OSAs can also be used as mill make up water when available. For the underground operation, a fresh water supply line (HDPE) will be constructed from the water treatment plant to the underground yard. A water storage tank will then provide clean water down the decline for underground mining equipment and water use at the underground yard.

The site is connected to the town of Kershaw municipal water system for potable water supply.

19 Market Studies and Contracts

The market for gold doré is well established. Market predictions and discussions for gold are beyond the scope of this document. The impacts of gold price volatility on the mine plan and process operation are well understood.

A contract is in place with Metalor USA Refining Corporation, located in North Attleboro, Massachusetts for the refining of doré bullion (Metalor). This company is a subsidiary of the Metalor Group, previously Métaux Précieux SA Metalor, which was a well-known and established precious metal refiner founded in 1852. The Metalor Group is nowadays a subsidiary of Japan's Tanaka Kikinzoku Group and is headquartered in Marin, Switzerland.

The contract has a one year term starting with an Effective Date of January 30, 2017, subject to termination by either party. This contract also sets a range of prices and surcharges for refining the doré under terms and conditions which generally comply with industry norms. It is assumed that these contract terms will be renewed through the LoM operation without changes or will be negotiated with a new refiner if necessary.

19.1 Contracts and Status

The Qualified Person is not aware of any forward sales or hedging contracts for Haile metal production. Other contract details are provided by OceanaGold at the time of writing this report as follows:

- Mine Fleet
    - Supplier: Caterpillar
    - Headquarters Location: Peoria, Illinois
    - End Date of Contract: We have a Master Finance Lease Agreement with Caterpillar Financial Services Corporation (Headquarter in Nashville, Tennessee). The current Schedule No. 2, dated August 10, 2016, is for a 60-month lease term, ending August 10, 2021.
- Grid Power
    - Supplier: Lynches River Electric Cooperative
    - Headquarters Location: Pageland, South Carolina
    - End Date of Contract: December 31, 2030
- Bulk Diesel
    - Supplier: TBD once fueling facilities are installed at the Project
- Cyanide
    - Supplier: The Chemours Company FC, LLC (80% supplier) / Cyanco International, LLC (20% supplier)
    - Headquarters Location: Chemours – Wilmington, Delaware / Cyanco – Pearland, Texas (parent company Cyanco Corporation Headquarters in Reno, Nevada)
    - End Date of Contract: Chemours – December 31, 2018 / Cyanco – November 15, 2018
- SAG Mill Grinding Media
    - Supplier: Magotteaux, Inc. (80% supplier) / ME Global Inc., d/b/a ME Elecmetal (20% supplier)
    - Headquarters Location: Magotteaux – Principal Place of Business in Franklin, Tennessee / ME Elecmetal – Minneapolis, Minnesota

  - End Date of Contract: Magotteaux – 1 August 2018 / ME Elecmetal – September 15, 2018
- Explosives (ANFO and/or Emulsion)
  - Supplier: DYNO Nobel, Inc.
  - Headquarters Location: Salt Lake City, Utah
  - End Date of Contract: April 15, 2018

20 Environmental Studies, Permitting and Social or Community Impact

Haile's current mine plan is based on construction, mining operation, closure, and reclamation of eight open pits, with three of those pits being left as pit lakes (Champion, Small and Ledbetter) and one as a partial pit lake (Snake).

In order to update the current mine plan, it is anticipated that a modified 404 Permit issued pursuant to the Clean Water Act will be required from the United States Army Corp of Engineers (USACE), which has jurisdictional responsibility for all Waters of the United States. It is also assumed that the USACE will again be the lead federal Agency for purposes of the National Environmental Policy Act (NEPA). The level of anticipated impact to the environment from the updated mine plan will determine the appropriate level of review under NEPA. Various permitting approvals/certifications will also be required from the South Carolina Department of Health and Environmental Control (SC DHEC), including modification of Haile's Mine Operating Permit and Reclamation Plan, TSF Permit, Air Permit, NPDES permits and 401 Water Quality certification. In addition, there will be approvals required from other federal and state agencies, including: United States Environmental Protection Agency (EPA), United States Fish and Wildlife Service (US FWS), South Carolina Department of Natural Resources (SC DNR), South Carolina State Historic Preservation Office (SC SHPO), South Carolina Department of Transportation (SC DOT) and Catawba Indian Nation. NEPA also allows non-governmental organizations (NGOs) and other interested parties an opportunity for review and comment on the anticipated impacts.

In late 2014, HGM. received all of the major permits required for construction and operation of the current mine plan, including: Clean Water Act 404 Dredge and Fill Permit, Mine Operation Permit, 401 Water Quality Certification, TSF Dam Permit, North Fork Dam Permit, Air Permit (to construct), and various NPDES Permits (wastewater treatment and stormwater). A non-government organization appealed the Mine Operating Permit but this was satisfactorily resolved in January of 2015.

Haile is unique in that it occurs wholly on private land owned by HGM and does not impact federal/public (BLM or USFS) lands that would be subject to projected modifications from these surface management agencies

There is a significant amount of existing background and environmental baseline data available for the Project. This data continues to be collected and reported to the regulators as part of operational controls. Additional data and environmental studies (Section 20-3) will be of technical interest to the federal and state agencies in evaluating a request to expand the current mining operation.

Permits currently held by HGM may be kept, modified, terminated, or replaced during the expansion process.

Table 20-1 is a summary of the current HGM permits.

Table 20-1: Mine Permits

Agency	Permit/Authorization Number	Date Received	Description
US Army Corps of Engineers	404 Permit – SAC-1992-24122-4IA	27 Oct, 2014	Permit to affect wetlands and streams per the approved Mine Plan.
U.S. Army Corps of Engineers	Permit 2004-1G-157	16 Oct, 2007	Permit to fill a portion of the old North Fork Creek
Mine Safety and Health Administration (MSHA)	MSHA ID: 38-00600	5 Feb, 2010	Operate mine within MSHA standards
Federal Communications Commission	Call Sign: WQJB814	18 Jul, 2008	Base station frequency, ten local frequencies
South Carolina Department of Health and Environmental Control (SCDHEC), Bureau of Water	401 Water Quality Certification	23 Oct, 2014	Water Quality certification to construction and operate a gold mine on HGMC, Camp Branch Creek, unnamed tributaries and adjacent wetlands.
SCDHEC, Division of Mining and Solid Waste Management	Mining/Operating Permit No. I-000601	5 Nov, 2014	Mine Operating Permit – Regulation of closure and reclamation.
SCDHEC, Bureau of Solid and Hazardous Waste Management	Permit No. SCD987596806	12 Apr, 1993	Conditionally large quantity generator
SCDHEC, Industrial Wastewater (IW) Permitting Section	National Pollutant Discharge Elimination System Discharge Permit No. SC0040479	7 Oct, 2013	Permit to discharge treated water from the mine operation / reclamation areas from Outfall 003.
SCDHEC, Industrial Wastewater Permitting Section	WTR-Wastewater Construction Permit Permit No. 19852-IW	30 Jan, 2015	Permit to construct sewer lines
SCDHEC, Bureau of Water, Industrial, Agricultural, and Storm Water Permitting Division	Dams & Reservoirs Safety Permit 29-0007 (Issued October 7, 2013)	7 Oct, 2013	Dam Safety Permit – Significant Hazard (Construction). Stability during earthquake-induced ground motion was evaluated by SCDHEC prior to issuance of the TSF construction permit. SC DHEC completed evaluation of the seismic stability study pursuant to the International Commission of Large Dam (ICOLD) design and performance standards.
SCDHEC, National Pollutant Discharge Elimination System (NPDES) Program, Water Facilities Permitting Division	General Permit for Stormwater Discharges for Small and Large Construction (Activities Permit) SCR100000	1 Jan, 2013	Discharge of stormwater in connection with construction of structures not covered under the Industrial General Permit – requires submittal of Storm Water Pollution Prevention Plan (SWPPP) and public notice prior to construction
SCDHEC, NPDES Program, Water Facilities Permitting Division	Stormwater discharges associated with industrial activity SCR000000, Permit No. SCR004763	1 Jan, 2011	Discharge of stormwater in connection with industrial activities, Industrial General Permit

Agency	Permit/Authorization Number	Date Received	Description
SCDHEC, Office of Environmental Quality, Bureau of Air Quality	Bureau of Air Quality, State Construction Permit No. 1460-0070-CA	4 Oct, 2013	Authorizes construction of the proposed facility and equipment specified in HGM's application for a Department of Army permit; a permit to operate also is required.
Lancaster County Council	Floodplain Development Permit June 27, 2013	27 Jan, 2013	Floodplain Administrator oversees and implements the provisions of the Flood Damage Prevention Ordinance.
Lancaster County Council	Ordinance 2013-1207	1 Jan, 2015	Rezoned the Haile property within the permit boundary to the M, Mining District designation.
SCDOT	177006	14 Jan, 2015	Encroachment Permit

Source: OceanaGold

20.1 Required Permits and Status

Table 20-2 is a summary of the permits necessary to start modification of the Open Pit Mine Plan, initiation of the Underground Mine Plan, or installation of additional equipment for the Process Plant optimization. HGM will submit permit applications at the conclusion of the required Environmental Studies.

Table 20-2: Environmental Permits

Application	Permit Required	Regulatory Agency
Open Pit Mining Modification	Mining Operating Permit No. I-000601	SCDHEC, Division of Mining and Solid Waste Management
	Fill or modify wetlands	US Army Corps of Engineers
	Surface water diversion	SCDHEC, Bureau of Water
	Installation of HGMC diversion dam	SCDHEC, Bureau of Water, Dams & Reservoirs Safety
	Expand current Mine Permit Boundaries	SCDHEC, Division of Mining and Solid Waste Management with cooperation from US EPA, US FWS, SC SHPO, and SC DNR, and Catawba Indian Nation
	New and modified existing Potentially Acid Generating (PAG) Storage Facilities and Collection Ponds	SCDHEC, Division of Mining and Solid Waste Management with cooperation from US EPA, US FWS, SC SHPO, and SC DNR, and Catawba Indian Nation; and SCDHEC, Bureau of Water
	Additional Mine Equipment incl. Haul Trucks, Excavators, Water Carts, Graders, Light Trees, and Generators	SCDHEC, Office of Environmental Quality, Bureau of Air Quality
Underground Mine	Mining Operating Permit	SCDHEC, Division of Mining and Solid Waste Management
	Ventilation Shaft Air Permit	SCDHEC, Office of Environmental Quality, Bureau of Air Quality
	Cement Plant Permit	SCDHEC, Office of Environmental Quality, Bureau of Air Quality and SCDHEC, Division of Mining and Solid Waste Management
	Groundwater disturbance	SCDHEC, Bureau of Water
	Underground Depressurization Water Wells	SCDHEC, Division of Mining and Solid Waste Management and SCDHEC, Bureau of Water
	Additional Mine Equipment incl. Haul Trucks, Excavators, Water Carts, Graders, Light Trees, and Generators	SCDHEC, Office of Environmental Quality, Bureau of Air Quality
Process Plant Optimization	Additional Process Plant Equipment – 200 t RoM Bin, Pebble Grinding, Tertiary VertiMil Circuit, Regrind VertiMil Circuit, and Floc Plant	SCDHEC, Office of Environmental Quality, Bureau of Air Quality
	Modified Equipment Run Rate	SCDHEC, Office of Environmental Quality, Bureau of Air Quality and SCDHEC, Division of Mining and Solid Waste Management
	Expand Tailing Storage Facility surface area	SCDHEC, Division of Mining and Solid Waste Management
	Modified Tailing Storage Facility Dam Height	SCDHEC, Bureau of Water, Dams & Reservoirs Safety
	Re-route Highway 601	SC DOT
	Modification to Earnest Scott and Duckwood Roads	Lancaster County
Extended Support Services	Extended Operation of Water Treatment Plant	SCDHEC, Bureau of Water
	Extended Site-wide Monitoring	SCDHEC, Division of Mining and Solid Waste Management

Source: OceanaGold

20.2 Environmental Study Results

Table 20-3 is a summary of the environmental studies initiated by HGM to support the approval process.

Table 20-3: Environmental Studies

Study	Scope of Work
Air Emissions	Assess impact to air pollution loading based on additional operating conditions and new equipment on active point sources – engine exhausts, conveyor drop points, discharge stacks, ventilation shafts, dust controls, blast gasses, cement plant, etc.
Aquatic Resources – plants and animals	Create Carolina Heel Splitter mussel fish and macroinvertebrate studies to assess impact to species over LOM.
Cultural & Historical Resources	Review and assess potentially impacted cultural and historical sites from surface disturbances. Relocate any potential gravesites.
Economic Impact to Local Community	Social and economic effect on the state and local economy – effect on local businesses, wages, local resources, emergency services, and external jobs.
Geochemistry Analysis	Update OMP for PAG material placement, ARD control, and underground cut and fill practices.
Hydrology Assessment	Identification of direct and indirect impacts on local wetlands, wells, and streams.
Impacted Wetland Assessment	Assess potentially impacted wetland vegetation around Unnamed Tributary, and newly acquired Jones and Gregory properties.
Reclamation Plan	Pit closure and remediation plans, surface controls, revegetation plans, stormwater control plans, surface run-off plans, timelines, and sequence for surface reclamation activities.
Surface Water Impacts	Assess impact to surface water flows – volume, dissolved oxygen, chemistry, pH and conductivity. Assess drainage patterns and develop recommendations for additional monitoring and measurement stations, if required. Create Stormwater plans for sediment ponds, location for BMP[1] devices, assessment locations, and site controls.
Terrestrial Resources – Plant Life	Perform terrestrial plant evaluations, specifically in impacted areas and areas of significant disturbance.
Terrestrial Resources – Wildlife	Perform seasonal terrestrial studies on migratory endangered wildlife, such as species of bats and raptors.
Traffic / Road Impacts	Perform a traffic study and predict road patterns and potential impacts.
Vibration Analysis	Develop vibration predictions based on underground blasting and changes in surface contours and geological morphology.
Wetland Delineation	Create surface maps with jurisdictional delineations. These will include necessary 50-ft (15.25-m) offset boundaries.
Wetland Measurement and Marking	Set the visual markers around the 50-ft offset. Measure the final size of wetland delineation.

Source: OceanaGold

[1] Best Management Practice Devices – 1) Erosion Prevention – slope surfaces, seeding, and erosion controls; and 2) Sediment Control - check dams, sediment dams, sediment ponds, and silt fencing.

20.3 Environmental Issues

As required by Haile's Mine Operating Permit, a US$65 million Reclamation Trust Agreement is currently in place between HGM and SCDHEC. It is to provide financial assurance to the State of South Carolina that funds will be available (in the event of default by HGM) to implement and complete the Reclamation Plan and for implementing, maintaining, repairing, or enhancing any aspect of reclamation, closure, and post closure activities. The financial assurance is in the form of surety bonds and an interest-bearing trust account.

It is anticipated that an increased financial assurance amount will be required by SCDHEC as a result of an expanded mining operation.

21 Capital and Operating Costs

21.1 Capital Cost Estimates

A summary of the total capital cost is provided in Table 20-1; the basis of the capital cost estimate is discussed below.

Table 21-1: Total Capital Cost Summary

Description	Unit	Expansion	Sustaining	Total
Open Pit	$000's	59,928	73,068	132,996
Underground	$000's	55,361	25,696	81,057
Process Plant	$000's	58,881	25,570	84,451
Infrastructure	$000's	64,822	119,551	184,373
Subtotal Capital Before Contingency	**$000's**	**238,991**	**243,885**	**482,876**
Contingency	$000's	15,725	2,200	17,925
Total Capital Before Revenue Credit	**$000's**	**254,716**	**246,085**	**500,800**
Net Preproduction Revenue Credit	$000's	(69,153)	-	(69,153)
Total LoM Net Capex	**$000's**	**185,563**	**246,085**	**431,648**

Source: SRK

21.1.1 Basis for Capital Cost Estimates

The capital cost estimates throughout this section has a base or effective date of Quarter 1 2017. All values are in United States dollars (US$), and no foreign currencies have been considered in the estimates.

Open Pit

Projected capital costs for open pit mining have been developed based on mine production schedules over more than 15 years to achieve a production rate of 4.0 Mtpa ore.

Total mining equipment capital costs over the mine life are estimated to be US$142.2 million. The major component of this cost is the incremental increases or replacements required in mining equipment fleets that are already established at site. These LOM capital costs are summarized in Table 21-2. The other category includes miscellaneous, technical, inventory and salvage.

Table 21-2: Open Pit Mining Capital Cost Summary

Description	Total Cost (US$000's)
Mobile Equipment	125,662
Mine Support Equipment	12,194
Miscellaneous	4,319
Total	**142,175**

Source: OceanaGold

Underground

Projected capital costs for underground mining have been developed based on mine production schedules over more than five years to achieve a production rate of 700 ktpa ore and are summarized in Table 21-3.

Total development-related expansion capital costs are estimated to be US$36.4 million which includes US$4.3 million of capitalized preproduction costs incurred in Q4 2020 before the UG operation achieves commercial production at its designed capacity.

Table 21-3: Underground Capital Cost Summary

Description	Total Cost (US$000's)
Capital Development	35,684
Mobile Equipment	22,612
Sustaining Maintenance	2,496
Other	15,881
Capitalized Operating Cost	4,383
Total	**81,057**

Source: SRK

Process Plant

M3 Engineering & Technology Corporation (M3) performed a prefeasibility-level engineering trade-off study to facilitate estimation of capital and operating cost components to be considered when evaluating an expansion of the Haile plant.

The M3 Estimated was valid for AACEI class 4 accuracy to ±20%. The cost for 4 Mtpa was US$95.55 Million inclusive of indirects, Engineering, Procurement and Construction Management (EPCM) (16.8%) and contingency (20%), etc. The M3 capital cost estimate was adjusted after review by OceanaGold. After adjusting the direct cost and applying lower contingency (15%) and indirect & owner's costs by factor (25%) a forecast of the plant expansion capital cost of US$67.63 Million, as shown in Table 21-4, was used for the financial evaluation.

Table 21-4: Process plant capital cost adjustments

Description	Total (US$000's)
M3 Estimate	95.55
M3 Estimate less contingency	79.63
Defer equipment purchase for 1 (of 2) Secondary Crushers	1.57
Modifications pre-completed in 2017 (e.g. pebble crusher)	2.49
Smaller Tertiary Tower mill	2.84
Extra Rougher Flotation Cell not required	0.77
Replace Concentrate CIL with Leach tanks only	6.22
Force flow thru upgraded existing Float Tails Thickener	2.52
Extra Elution capacity NOT required	0.57
Extra Elution capacity NOT required	1.12
Defer Tails pumping upgrade	2.38
Rationalise E-Room design & use existing MCC space	0.30
Total OceanaGold Modifications	20.78
Total OceanaGold estimated Capital	58.85
Total OceanaGold with contingency (15%)	67.63

Source: OceanaGold

The capital costs required to expand to a 4.0 Mtpa capacity processing facility are at an optimal range of limit. In other words, this expansion is able to take advantage of many modest capital modifications due to conservative initial design.

Expanding the plant beyond this point will require more extensive capital changes with a high number of equipment modifications or replacements being required. Expansion beyond this level would require that the process be effectively duplicated in entirety and the nature of the project site will limit the footprint in which new facilities for the expansion could be installed.

Direct costs are estimated as the total installed cost of equipment. Freight is included as a direct cost.

Major equipment costs are derived from vendor quotations based on equipment data sheets. Some equipment costs are derived from selected vendor quotes and an extensive M3 database. Due to the recent history of purchased materials for Haile, valid data was extracted from these purchase orders with escalation factors applied where necessary based on vendor recommendations and experience.

Indirect costs are primarily factored costs of the installed directs and take into account items including contractor mobilization, engineering, construction management, spares, first fills, contingency and owner's costs, etc.

Existing equipment in the Haile mill is considered a "sunk" cost for this cost estimate. No allowances are considered for operational downtime related to construction activity.

Infrastructure

Infrastructure capital associated with the expanded project is estimated to be US$148.0 million for the LOM. A summary of the major items is shown in Table 21-5. Infrastructure capital has been estimated internally by OceanaGold or provided by external consultants. The major capital items are PAG storage, tailings storage facility expansion, water diversion channel and in pit dewatering.

The remaining capital items have been sourced from the 2016 Haile LOM plan. The major infrastructure items such as PAG expansion, tailings storage facility lifts, water management and open pit dewatering which were budgeted in the 2016 Haile LOM plan have been excluded as these items were re-estimated because designs have changed and footprint areas have expanded.

Table 21-5: Infrastructure Capital Summary

Description	Total Cost (US$000's)
PAG Expansion	53,213
TSF Expansion	63,381
Water Management	17,073
Grid Power Upgrade	3,050
Open Pit Dewatering Wells	11,250
Land Administration	4,575
Property Management	23,551
Vehicles	5,680
IT, Software and Furniture	2,599
Total	**184,373**

Source: OceanaGold

21.2 Operating Cost Estimates

A summary of the total operating cost is provided in Table 21-6; the basis of the operating cost estimate is discussed below.

Table 21-6: Total Operating Cost Summary

Description	Total Cost (US$000's)
Open Pit	771,961
Underground	120,133
Processing	596,491
General & Administration	190,313
Selling/Refining	6,213
Indirect Costs	127,837
Total	**1,812,948**

Source: SRK

21.2.1 Basis for Operating Cost Estimates

The operating cost estimates throughout this section has a base or effective date of Quarter 1 2017. All values are in United States dollars (US$), and no foreign currencies have been considered in the estimates.

Open Pit

Projected operating costs for mining have been developed based on mine production schedules over more than 15 years to achieve a production rate of 4.0 Mtpa ore.

The overall fleet operating costs are built up from the hourly rates for each equipment type times the fleet operating hours for that type. The unit rates have been developed from first principles and benchmarked against existing OceanaGold operations (including current site costs) and North American operations. Labor costs on an annual basis are built up by first estimating the manpower required based on the units of equipment and the roster plus the requirement for management and technical staff.

All mining costs incurred during the 9-month preproduction period from January 1 through September 30, 2017 are recategorized as capital in the TEM and not included in the RoM operating cost metrics.

The average cost of mining rock over the mine life is US$1.48/t and the cost by activity is presented in Table 21-7.

Table 21-7: Open Pit Mining Cost Summary

Description	US$/t mined (1)	US$000's
Drilling	0.11	57,717
Blasting	0.15	79,527
Loading	0.18	95,651
Hauling	0.58	311,513
Ancillary	0.20	104,958
Grade Control	0.03	17,646
Mining Overheads	0.06	30,321
Salaried Labor	0.17	93,583
Dewatering	0.01	6,471
Total OP Mining Cost	**1.49**	**797,388**
Capitalized Operating Cost	1.74	(25,428)
Total OP Mining Opex	**1.48**	**771,961**

Source: OceanaGold
(1) Ore + Waste

Underground

Projected operating costs for underground mining have been developed based on mine production schedules over more than five years to achieve a production rate of 700 ktpa ore.

The average cost of underground mining during steady state operations starting in Q1 2021 (Q4 preproduction costs are excluded) and lasting over a 5-year mine life is US$39.11/t and the cost by activity is presented in Table 21-8.

Table 21-8: Underground Mining Cost Summary

Description	US$/t Ore	US$000's
Production Costs (incl. waste)	8.14	25,366
Backfill Cost	5.61	17,494
Mine Services Allocation	4.02	12,533
Labor Allocation	22.18	69,124
LoM UG Mining Costs	**$39.96**	**$124,516**
Capitalized UG Preproduction Costs	(1.41)	(4,383)
Total Underground Mining Opex	**39.11**	**$120,133**

Source: SRK

Process Plant

The power cost component of the estimate was developed from a load list for the proposed expanded plant with estimated individual installed motor sizes nominated and running load and diversity factors applied. The current unit energy cost rates in the existing power supply agreement with the power supplier to the current operation (Lynches River Authority) were used.

Labor costs were developed based on a slightly revised staffing plan for the expanded plant due to the addition of some extra equipment in the modified flow sheet but used current labor rate schedules.

The reagent and grinding media consumption estimates are based on forecasts used in the current Haile LOM plan, adjusted as needed based on vendor advice for the revised flowsheet in the expanded plant.

The processing plant is operating in the commissioning and ramp up phase and as such there is insufficient data to validate LOM plan consumption estimates at the time of compilation of this report.

Similarly, grinding media and reagent unit costs were based on current Haile supply contract rates for consumables delivered to site for the current operation.

Crusher and mill liner replacement costs are based on vendor budget prices or the value of recently placed orders for linings for existing crushing and grinding equipment items. Liner usage estimates were based on forecasts used in the current Haile LOM plan for existing equipment or based on OEM advice for the equipment added to the flowsheet in the expanded plant.

Maintenance materials cost predictions are based on a factor applied to the actual installed direct costs for the existing plant equipment and the direct costs from the capital cost estimates for the additional areas added to the plant for the expansion.

Miscellaneous costs cover assay laboratory charges assigned to the process plant and other minor ad-hoc expenses such as software license and lease fees, technical consultancy services, development test work and advisors fees, etc.

These cost estimate are summarized in Table 21-9.

Table 21-9: Summary of Process Operating Costs

Description	$/t milled	US$000's Per Annum
Power	2.22	8,870
Labor	1.50	6,010
Grinding Media	1.45	5,805
Reagents	2.20	8,807
Crusher Liners	0.06	240
Mill Liners	0.66	2,638
Maintenance Materials	1.66	6,640
Miscellaneous	0.40	1,612
Assay	0.19	749
Total	**10.34**	**41,370**

Source: OceanaGold

Selling and Refining

Sales are refining charges are listed below.

- Au deduction 0.004%
- Ag deduction 1.000%
- Treatment charge of US$0.70 per troy ounce
- Transport charge US$750 for pick up and US$0.23 per troy ounce

General and Administration

General and administration costs have been sourced from the 2016 Haile LOM plan and are summarized in Table 21-10. When process plant throughput is expanded to 4.0 Mtpa and Underground operations reach steady state, general and administration costs increase to US$13.7 million per annum. Once Underground operations are complete, general and administration costs reduce to US$12.7 million per annum.

Table 21-10: General and Administration Cost Summary (US$000's)

Description	2017 to 2019	OP + UG 4.0 Mtpa	OP Only 4.0 Mtpa
General Overheads	2,128	3,922	3,622
Commercial	581	676	676
Safety	2,168	2,661	2,361
HR General	725	971	971
Information Technology	747	899	849
Supply Chain & Procurement	1,121	1,412	1,212
Environmental Compliance	2,100	2,751	2,751
Mine Site Overheads	4,200	445	445
Total	**9,990**	**13,737**	**12,737**

Source: SRK

Indirect Costs

Indirect costs are estimated to be US$127.8 million for the LOM. These costs have been sourced from the 2016 Haile LOM plan and or estimated by OceanaGold. Indirect costs include rehabilitation, closure, mitigation and finance lease payments and are summarized Table 21-11.

Table 21-11: Indirect Cost Summary

Description	Total Cost (US$000's)
Rehabilitation and Closure	60,000
Endowment Liability – State of South Carolina	9,686
Principal/Interest Payment – Capital Leases	32,197
Regional Business Unit Costs	25,954
Total	**127,837**

Source: SRK

22 Economic Analysis

22.1 Principal Assumptions and Input Parameters

The indicative economic results summarized in this section are based upon work performed by SRK and OceanaGold in 2017. They have been prepared on both a periodic monthly/quarterly/annual format and an annual format. The metrics reported in this volume are based on the annual cash flow model results. The metrics are on both a pre-tax and after-tax basis; a 100% equity basis with no Project financing inputs; and are in Q1 2017 U.S. constant dollars.

Key criteria used in the analysis are discussed in detail throughout this section. Principal Project assumptions used are shown summarized in Table 22-1.

Table 22-1: Basic Model Parameters

Description	Value
TEM Time Zero Start Date	January 1, 2017
OP Operations Commercial Production Start	October 1, 2017
UG + Mill Expansion Development Start	January, 2020
UG + Mill Expansion Commercial Production Start	January, 2021
Open Pit Mine Life	16 years
UG Mine Life	5 years
Discount Rate	MOP @ 5%

Source: SRK, 2017

All costs incurred prior to January, 2017 are considered sunk with respect to this analysis.

The selected Project discount rate is 5% as directed by OceanaGold and the valuation uses middle-of-period discounting. A sensitivity analysis of the discount rate is discussed later in this section.

Foreign exchange impacts were deemed negligible as most, if not all costs and revenues are denominated in US dollars.

When Haile commences commercial production, the capitalization of certain mine construction and operation costs will cease and costs will either be attributed to inventory or expensed in the period in which they are incurred, except for capitalized costs related to property, plant and equipment additions or improvements, open pit stripping activities that provide a future economic benefit, and exploration and evaluation expenditure that meets the criteria for capitalization. It is also at this point that depreciation and amortization of previously capitalized costs commences. Until the date of commencement of commercial production, any revenues (net of matching operating costs) recognized from the sale of gold are credited as a reduction to development costs capitalized (Net Preproduction Revenue Credit). It is assumed in the TEM that the starter Project will commence commercial production on October 1, 2017 and that the UG mine/mill expansion project will reach commercial production on January 1, 2021. The actual timing for commencement of commercial is a matter of management judgement and is not known definitively.

22.2 Cashflow Forecasts and Annual Production Forecasts

TEM inputs/results for the Cashflow Forecasts are summarized on a LoM basis in this section while a full LoM annual cash flow forecast is presented in Appendix B.

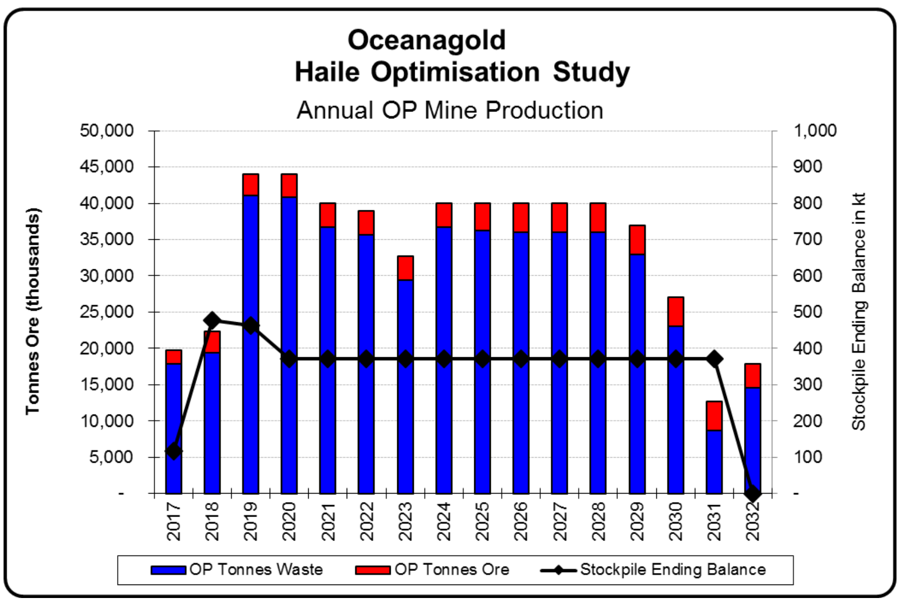
22.2.1 Mine Production

Table 22-2 is a summary of the estimated mine production over a 16-year mine life for the combined open pit/underground operations. Ore mined refers to Proven and Probable Mineral Reserves. Figures 22-1 and 22-2 show LoM production by year for OP and UG operations.

Table 22-2: Life-of-Mine Production Summary

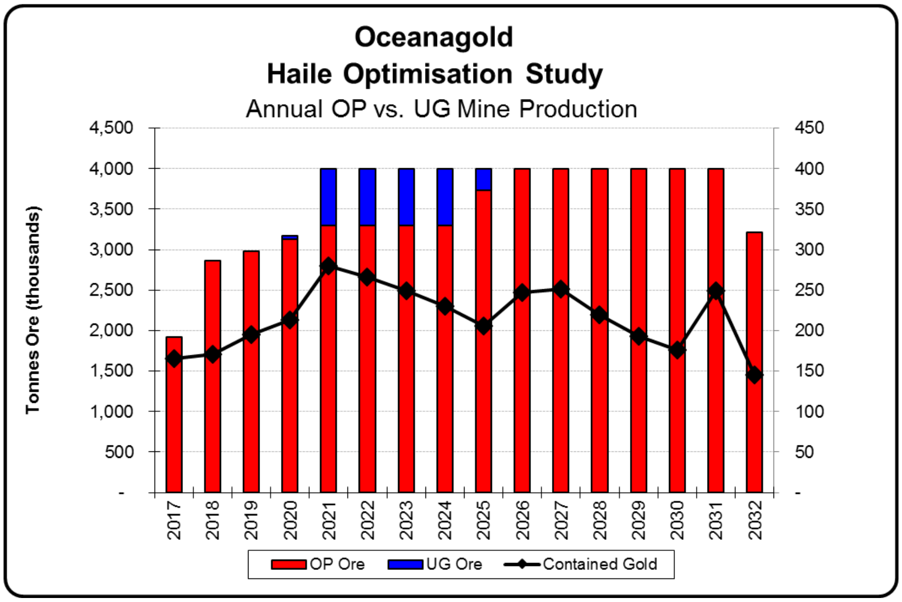
Description	Value	Units
OP Ore Mined	55,039	kt
OP Waste Mined	481,174	kt
OP Total Material Mined	536,213	kt
OP Mined Gold Grade	1.71	g/t
OP Contained Gold	3,018	koz
OP Stockpile Ore Ending Balance (Max)	526	kt
OP Stockpile Grade, Gold	2.02	g/t
OP Stockpile Contained Gold	34	koz
UG Ore Mined	3,116	kt
UG Mined Gold Grade	4.38	g/t
UG Contained Gold	439	koz
Total Ore Mined	**58,155**	**kt**
Waste Mined	481,174	kt
Total Material Mined	536,213	kt
Maximum Daily Mining Capacity	122,222	t/d
Total RoM Grade	**1.85**	**g/t**
Total Contained Gold	**3,457**	**koz**

Source: SRK, 2017



Source: SRK, 2017

Figure 22-1: Annual Open Pit Mine Production



Source: SRK, 2017

Figure 22-2: Annual Open Pit and Underground Ore Production

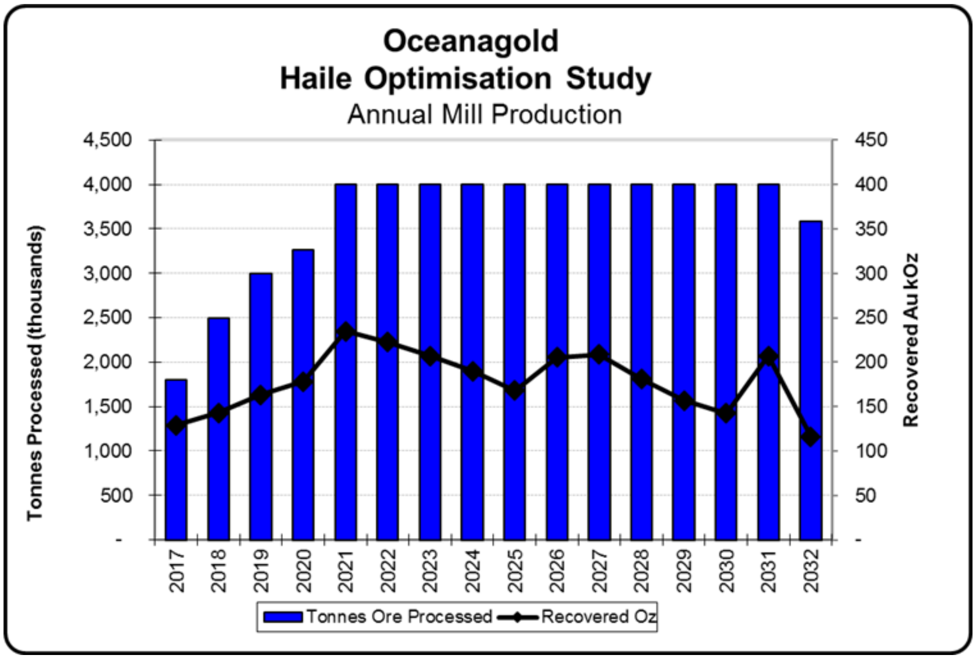
22.2.2 Mill Production

A summary of the estimated process plant production for the Project is contained in Table 22-3 for a 16 year operating life. Figure 22-3 shows LoM production by year with the mill expansion coming on stream in 2021 at 10,959 tonnes per day capacity or 4Mtpa. Ore processed refers to Proven and Probable Mineral Reserves.

Table 22-3: Life-of-Mine Process Production Summary

Description	Value	Units
Total Ore Processed	58,155	kt
Daily Process Capacity (Before Expansion)	6,849	t/d
Daily Process Capacity (After Expansion)	10,959	t/d
Processed Grade	1.85	g/t
Contained Gold	3,457	koz
Recovery	82.5%	%
Recovered Gold	2,854	koz

Source: SRK, 2017



Source: SRK, 2017

Figure 22-3: Annual Process Plant Production

22.2.3 Revenue

Gold pricing assumptions used in the economic analysis include a constant base case (aka Neutral Case) LoM gold price of US$1,300/troy oz which was provided by OceanaGold's Business Planning group for 2017 budgeting purposes for their worldwide operations.

Doré refining/freight costs, as previously discussed in Section 19, are as follows:

- 100.0% payable Au;
- US$0.70/troy oz Au treatment charge; and
- US$0.23/troy oz Au + US$750/shipment freight costs.

Metallurgical and commissioning data to date shows that the Project will generate a small annual silver by-product credit even though silver content was not included in current the Mineral Resource or Reserve estimates.

The silver by-product credit in the TEM is calculated by using a constant silver price of US$19/troy oz; a 1.5:1 Ag/Au oz ratio; and 70% recovery. The additional silver related doré refining costs are as follows:

- 99.0% payable Au;
- US$0.70/troy oz Au treatment charge; and
- US$0.23/troy oz Au freight cost.

The average annual net silver by-product credit of ~US$4.3 million (US$68.2 million over LoM) represents of 1.8% of revenue over LoM for the Project.

22.2.4 Operating and Capital Costs

Operating cost metrics in the TEM are reported on a RoM basis meaning that all costs and mined/processed tonnes starting from the assumed first day of commercial production (October 1, 2017 for the Project) through the end of operations in 2032 are included in the metrics. Conversely, all operating costs incurred during the 9-month preproduction period from January 1 through September 30, 2017 are recategorized as capital in the TEM and not included in the RoM operating cost metrics.

The total RoM operating cost unit rate of US$31.91/t processed is summarized inTable 22-4.

Table 22-4: RoM Operating Cost Summary

Description	US$000s	US$/t ore processed
OP Mining ($/t rock)	771,961	1.48
UG Mining ($/t ore mined)	120,133	39.11
Subtotal Mining	892,094	15.71
Processing	596,491	10.50
G&A Cost	190,313	3.35
Refining/Freight Costs	6,213	0.11
Indirect Costs	127,837	2.25
Total Operating Expense	**$1,812,948**	**$31.91**

Source: SRK, 2017

There are several important items in the Indirect Costs line of which are further detailed in Table 22-5, including US$60 million of concurrent and EoM closure/reclamation costs incurred.

Table 22-5: RoM Indirect Costs Summary

Description	US$000s	US$/t ore processed
Royalties	-	-
Reclamation/Closure Costs (1)	60,000	1.06
Endowment Liability - State of SC	9,686	0.17
Principal Payment/Interest - Capital Leases	32,197	0.57
Regional Business Unit Costs	25,954	0.46
Total Indirect Costs	**$127,837**	**$2.25**

Source: SRK, 2017

(1) Full LoM closure costs, including 4 years of post-closure costs, are included in this table for completeness. However, for this analysis post-closure costs are not considered in RoM metrics nor AISC.

Total LoM capital costs totaling US$501 million not including final closure/reclamation costs are summarized in Table 22-6. All closure/reclamation costs are included in Indirect Costs in the operating cost estimate. Total Development/Expansion capital costs of US$254.7 million including a 6.6% contingency are part of this total. The development/expansion capital estimate is further offset by a Net Preproduction Revenue Credit of US$69.2 million generated from gold sales made in the period of January 1 to September 30, 2017 just before the start of commercial production on October 1, 2017. The Net Preproduction Revenue credit calculation is shown in Table 22-7.

Table 22-6: Life-of-Mine Capital Costs (000's)

Description	Initial Capex	Sustaining Capex	Total
OP Mining	59,928	73,068	132,996
UG Mining	55,361	25,696	81,057
Plant Expansion	58,881	25,570	84,451
Infrastructure Expansion	64,822	119,551	184,373
Subtotal Capex Before Contingency	**$238,991**	**$243,885**	**$482,876**
Contingency	15,725	2,200	17,925
Contingency % of Capex	6.6%	0.9%	3.7%
Total Capex Before Revenue Credit	**$254,716**	**$246,085**	**$500,800**
Net Preproduction Revenue Credit	(69,153)	-	(69,153)
Total Net Capex	**$185,563**	**$246,085**	**$431,648**
Reclamation/Closure (1)	-	-	-
Total LoM Net Capex	**$185,563**	**$246,085**	**$431,648**

Source: SRK, 2017
(1) Part of Indirect Costs under Opex

Table 22-7: Net Preproduction Revenue Credit

Description	US$000s
Gross Preproduction Gold Revenue	126,472
Preproduction Silver By-Product Credit	2,460
Subtotal Preproduction Revenue	**$128,932**
OP Mining	(25,428)
Processing	(17,214)
G&A	(7,493)
Selling and Refining	(231)
Indirect Costs	(9,414)
Subtotal Capitalized Preproduction Costs	**($59,780)**
Total Net Preproduction Revenue Credit	**$69,153**

Source: SRK, 2017

An estimate of US$1.7 million of working capital is estimated for the first year of commercial production starting in August 2017. The assumptions used for this estimate are as follows:

- Accounts Receivable (A/R): 5-day delay;
- Accounts Payable (A/P): 30-day delay; and
- Consumable Inventory: 30-day supply

Annual adjustments to working capital levels are made in the TEM with all working capital recaptured by the end of the mine life resulting in a LoM net free cash flow (FCF) impact of US$0.

22.2.5 Economic Results

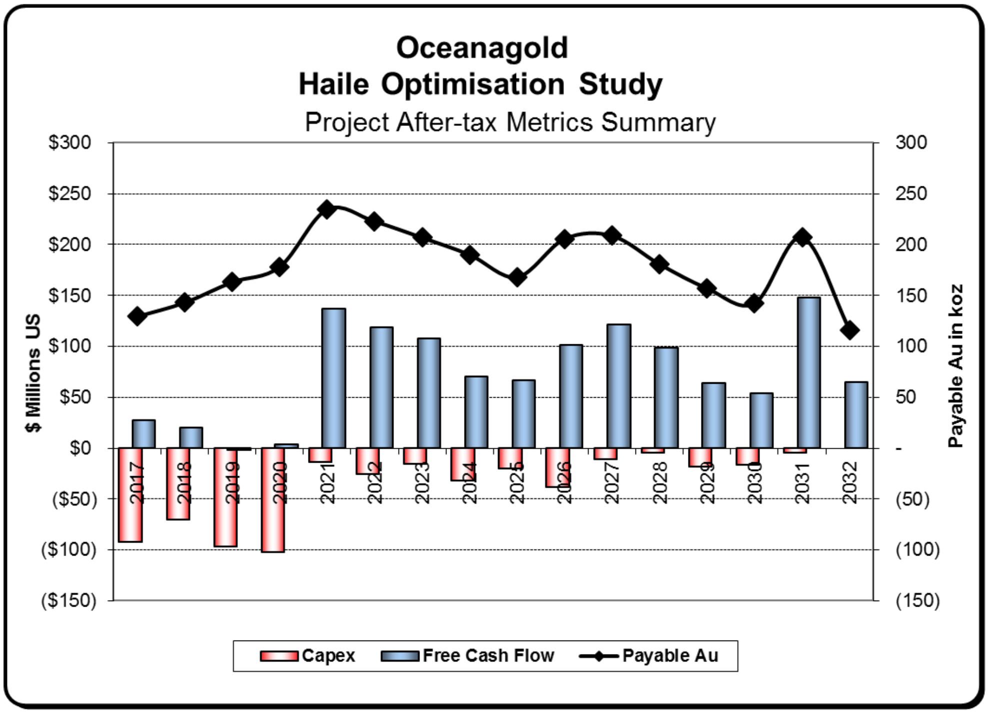
The TEM metrics are prepared on an annual after-tax basis, the results of which are summarized in Table 22-8. A full LoM annual cash flow forecast is presented in Appendix B. The results indicate that at a US$1,300/oz gold price the Project returns an after-tax NPV 5% of US$770.5 million. Note that because the project is valued on a total project basis with prior capital treated as sunk, and not by an incremental analysis of the UG and mill expansions, an IRR value is not relevant in this analysis. Figure 22-4 presents annual cash flow metrics versus recovered gold production and shows that the project does not generate much positive free cash flow in the first four years due to the level of capital expenditures being made during that time.

Table 22-8: Indicative Economic Results

Description	US$000s
Market Prices	
Gold (US$/oz)	$1,300
Payable Gold (koz)	2,854
Revenue	
Gross Gold Revenue	3,709,892
Silver By-Product Credit (@ US$19 / oz Ag)	68,280
Total Gross Revenue	**$3,778,172**
Operating Costs	
OP Mining	(771,961)
UG Mining	(120,133)
Processing	(596,491)
Site G&A	(190,313)
Selling/Refining	(6,213)
Indirect Costs	(127,837)
Total Operating Costs	**($1,812,948)**
Operating Margin (EBITDA)	**$1,965,224**
Taxes	
Income Tax	(219,459)
Total Taxes	**($219,459)**
Working Capital (1)	0
Operating Cash Flow	**$1,745,765**
Capital	
Development/Expansion Capital	(314,495)
Sustaining Capital	(246,085)
Total Capital	**($560,580)**
Metrics	
Pre-Tax Free Cash Flow	$1,404,644
After-Tax Free Cash Flow	$1,185,185
Pre-Tax NPV @: 5%	$925,348
After-Tax NPV @: 5%	$770,471

Source: SRK, 2017

(1) Includes US$1.7 million in 2017



Source: SRK, 2017

Figure 22-4: Project After-tax Metrics Summary

Table 22-9 shows the build-up of a RoM AISC of US$701/oz net of a US$24/oz silver by-product credit over the 16-year life of the Project.

Table 22-9: RoM AISC Contribution

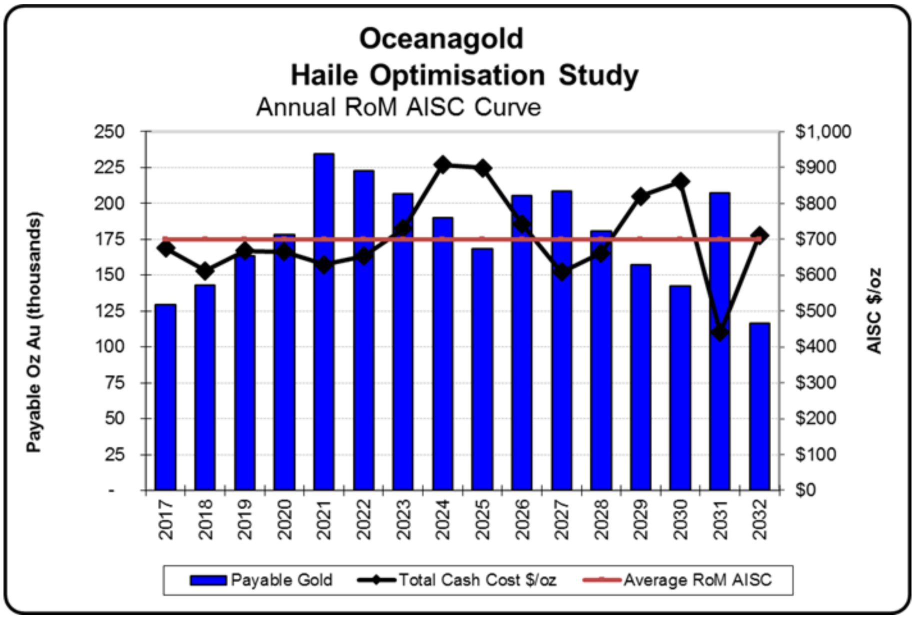
Total RoM Payable Gold Sales (Excl. Pre-production Sales) in koz		2,756
Description	**US$000s**	**US$/oz**
OP Mining	771,961	280
UG Mining	120,133	44
Processing	596,491	216
Site G&A	190,313	69
Selling/Refining/Freight	6,213	2
Direct Cash Costs Before By-Product Credit	**$1,685,112**	**$611**
Silver By-Product Credit	(65,820)	(24)
Direct Cash Costs After By-Product Credit	**$1,619,292**	**$587**
Royalties	-	-
Reclamation/Closure Costs (1)	41,200	15
Endowment Liability - State of South Carolina (2)	9,686	4
Principal Payment/Interest - Capital Leases (3)	32,197	12
Regional Business Unit Costs	25,954	9
Indirect Cash Costs	**$109,037**	**$40**
Sustaining Capex (4)	203,565	74
Total LoM AISC	**$1,931,894**	**$701**

Source: SRK, 2017
Per OGC Accounting Standards:

(1) On a cash basis, i.e., progressive/concurrent rehab where applicable is picked up in AISC as incurred, but post completion spend is not nor is and asset amortisation or unwind. As per Table 22-5, US$18.8 million of post closure costs are not included in RoM AISC.
(2) Included if cost is some form of provision for site employee retirement or other benefit payable later.
(3) Included if on a cash basis.
(4) OceanaGold treat TSF works completed with 5 years from commissioning as non-sustaining capital which for this analysis amount to US$14.4 million. If these costs are included, the LoM AISC would be US$706/oz.

Figure 22-5 shows the annual RoM AISC trend during the mine operations against an overall average RoM AISC of US$701/payable oz over the 16-year LOM at an annual production rate of 182 koz Au per year. The AISC variations are mainly driven by annual gold production levels and can range from US$440 to US$908 per oz in a given year but can be subdivided into three phases:

1. Starter OP AISC (Oct 2017 to 2020): US$669/oz Au;
2. UG+OP AISC (2021 to 2025): US$746/oz Au; and
3. Ending OP AISC (2026 to 2032 EOM): US$677/oz Au.



Source: SRK, 2017

Figure 22-5: Annual AISC Curve Profile

22.3 Taxes, Royalties and Other Interests

Due to the advanced state of the Project which is forecast to begin commercial operations in October 2017, a comprehensive taxation methodology was developed for the TEM with the assistance of OceanaGold's Project team's accountant. The project after tax forecast excludes any benefit associated with net operating losses (NOL) carried forward which are potentially available to the group. At the time of the report, the final position with regard to NOL carry forward had not been finalized. The assumptions used in the methodology are described in this section whereas a full taxation model is shown in Appendix C. The main taxation assumptions are as follows:

- Taxes calculation is on a periodic basis (monthly/quarterly/annual) for simplicity;
- Corporate Income Tax (CIT) rates are 35% for Federal and 5% for South Carolina;
- Federal taxable income is subject to Alternative Minimum Tax (AMT) of 20% per period;
- Net Operating Losses (NOL) may be carried forward indefinitely 20 years and can be used up to 100% of annual positive taxable income per period;
- Federal Depletion allowance is calculated for each period by determining the larger of:
  - 15% percentage depletion of Gross Income from Mining (subject to 50% of Net Income from Mining limit); and
  - Cost depletion basis amortizing US$642 million (C$856 million project purchase price converted at 0.75 C$/US$ exchange rate) over the LoM by unit of production method.
- Tax Depreciation allowance is calculated for each period by the following methods:
  - Undepreciated Initial Construction capex: Total of US$417 million incurred (sunk) before January 1, 2017 – 10-year straight line method beginning in January 2017;
  - Project-related Mobile Equipment capex: Total of US$151 million – 7-year straight line depreciation starting in period when cost is incurred;

- Project-related Development/Sustaining capex: Total of US$374 million (net of US$9.6 million of land acquisitions) – Unit of production method starting in January 2017 when incurred with the exception of plant expansion capital which starts depreciation in January 2021 after reaching commercial production; and
- All calculated depreciation remaining after final year of production is summed up and written off in the final year of production.

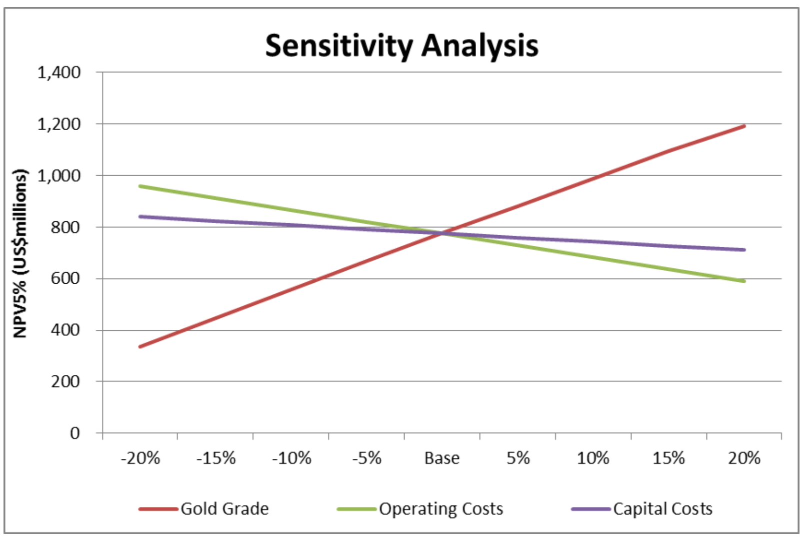
- Federal Domestic Productions Activities Deduction (DPAD) is calculated for each period by determining the larger of:
  - 9% of net taxable income amount; and
  - 50% of total mining salaries and wages cost.

There are no 3rd party government or private royalties or government severance taxes due on the Project during LoM.

22.4 Sensitivity Analysis

22.4.1 Operational Sensitivity

After-tax sensitivity analyses for key operational parameters are shown in Figure 22-6. Not surprisingly, the Project is nominally most sensitive to gold grade. The Project's sensitivities to capital and operating costs are similar but slightly more susceptible to operating costs.



Source: SRK, 2017

Figure 22-6: Operational Sensitivity Analysis

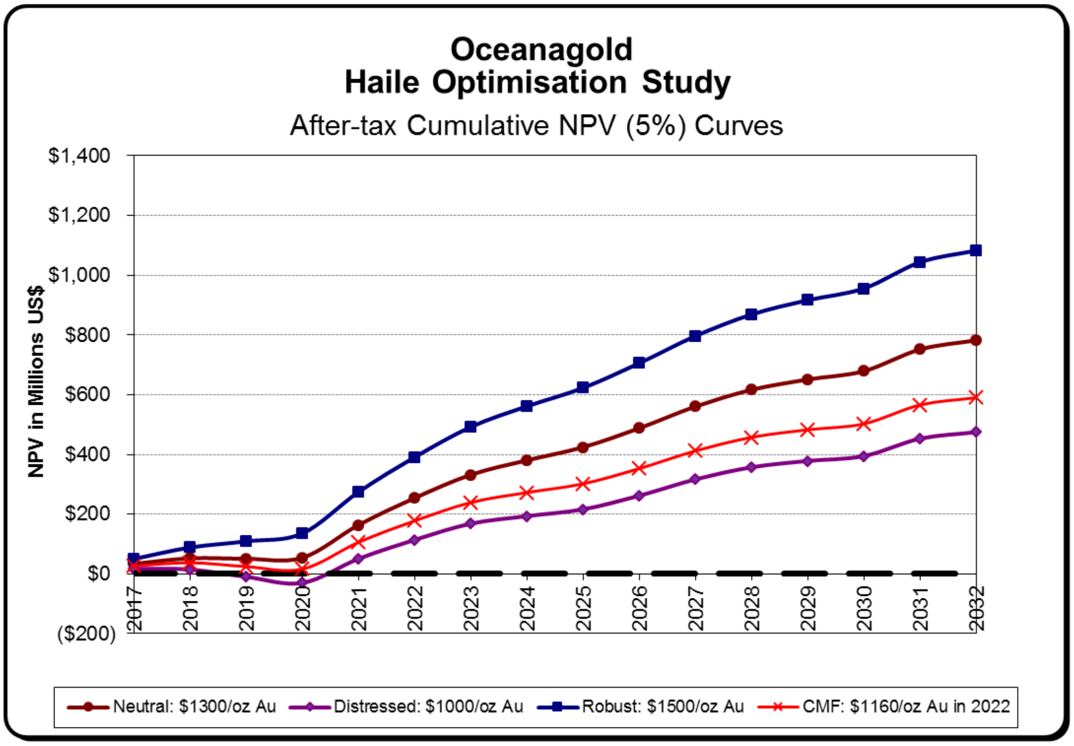
22.4.2 Gold Price Sensitivity

Additional gold price sensitivity analyses are shown in Figure 22 7 with after-tax Project NPV 5% at constant "Robust" prices of US$1,500/oz; constant "Distressed" price of US$1,100/oz; and a Consensus Economics' Q3 2017 Consensus Market Forecast (CMF), which shows a five year forecast with gold prices reaching US$1,160/oz in 2021 and constant from 2022 onward as follows:

- 2017: US$1,220

- 2018: US$1,220
- 2019: US$1,200
- 2020: US$1,200
- 2021: US$1,160
- 2022+: US1,160

All told, the after-tax Project NPV 5% changes approximately US$1.50 million for every US$1 change in gold price, either upwards or downwards.

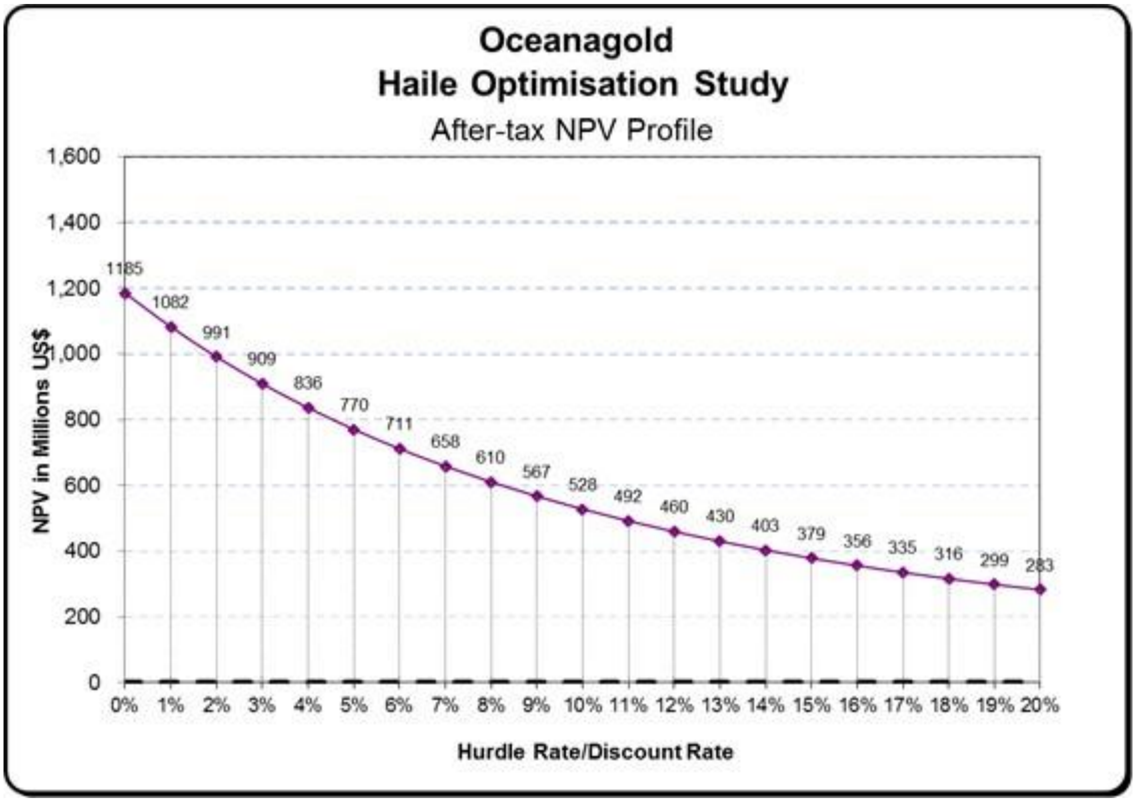


Source: SRK, 2017

Figure 22-7: Gold Price Sensitivity Analysis

22.4.3 Discount Rate Sensitivity

A sensitivity analysis of discount rates presented in Figure 22-8 shows that the Project as currently modelled would be NPV positive through a 20% discount rate.



Source: SRK, 2017

Figure 22-8: Discount Rate Sensitivity Analysis

23 Adjacent Properties

The Carolina terrane contains numerous gold mines and mining districts. Over 1,500 gold prospects have been documented. Most of these deposits were discovered in the 1800s. Significant gold deposits in South Carolina include the Haile, Ridgeway, Brewer, and Barite Hill Mines, Numerous quartz vein-hosted mines of the Gold Hill and Cid Mining Districts occur in North Carolina. Some gold deposits have similar geologic and mineralization features to Haile, and several are polymetallic with Cu, Ag, Pb and Zn.

23.1 Ridgeway Mine

The Ridgeway Mine is located 8 km east of Ridgeway, South Carolina and 40 km north of Columbia, South Carolina. Kennecott produced 1.5 million ounces (46,655 kg) of gold from 1988 to 1999 from two open pits in low-grade oxide and sulfide ore from siliceous deposits in the Richtex Formation (Gillon, 2000). The Ridgeway deposit has strong geological similarities to Haile (Gillon et al., 1995, 1998). The saprolite, volcanic and metasedimentary rocks are quartz-sericite-pyrite altered in mineralized areas. Post-mineral mafic and diabase dikes cross-cut the deposit, and are often accompanied by shearing and/or faulting. Gold grade is related to lithology, cleavage development, pyrite grain size and abundance, and silica content. Molybdenite is also associated with the mineralization.

The mine and mill had a production capacity of 13,608 tonnes per day. Ore was milled to minus 200-mesh then fed into a modified carbon-in-leach circuit. Carbon was stripped of gold, electroplated onto steel wool cathodes, and then transferred to electro-refining cells where gold was plated onto stainless steel plates. Mine closure and reclamation were successfully completed in the early 2000s.

23.2 Brewer Mine

The Brewer gold mine is located 12 km northeast of Haile in Jefferson County. The area was mined for iron prior to the Revolutionary War, before the first documented gold discovery in 1828 by Burrell Brewer.

Brewer rocks include schist, volcanics, and granite overlain by 40-60 feet of saprolite and sand. Gold mineralization is associated with quartz-sericite-pyrite altered schist, strong silicification and brecciation, and >2% pyrite. Gold ore was produced from a breccia body of hydrothermal origin and a related smaller body of fault-controlled ore. Pyrite content is generally 2-5%, unevenly distributed as aggregates and individual crystals in quartz veins. Gold grades were reported in the 1.41 g/t to 4.06 g/t range with associated silver, copper, tin, and bismuth. Brewer is classified as a high sulphidation breccia pipe hosted in the Persimmon Fork volcanics and may have deep porphyry roots.

Like Haile and other mines in the region, the mine produced gold intermittently, first as a placer, then as a surface and underground mine, and finally as a low-grade, heap leach operation in the 1980s.In 1987, Westmont Mining estimated a non-NI 43-101 compliant reserve for Brewer of 4.6 million tonnes grading 1.4 g/t gold (188,000 oz) (Scheetz, et al. 1991). The most recent production was from 1987 to 1995 by Westmont Mining/Costain Ltd Group. Ore was mined using conventional truck and loader open pit methods and ore was processed using cyanide leaching. In 1990, failure of an overflow pond during prolonged heavy rainfall released water containing sodium cyanide solution, copper, mercury, chromium, cobalt, nickel, and selenium into Lynches River. Brewer suffered from poor planning and closure, and became an EPA Superfund site in 1999.

23.3 Barite Hill Mine

The Barite Hill Mine is located 4 km southwest of McCormick, South Carolina. It is within the Lincolnton-McCormick Mining District, which includes other small mines and prospects of gold, silver, copper, zinc, lead, kyanite, and manganese.

The Barite Hill deposit was mined from 1989 to 1994 by Nevada Goldfields, Inc. The mine produced 59,000 oz of gold (1.8 million grams) and 109,000 oz (3.4 million grams) of silver, mainly from oxidized ore in the 20-acre (8 ha) Main Pit and the 4-acre (1.6 ha) Rainsford Pit. The mine used conventional open pit mining methods and an on/off pad heap leach process.

In June 1999, Nevada Goldfields Inc. filed for Chapter 7 bankruptcy, and abandoned the property. The property came under control of the South Carolina Department of Health and Environmental Control and the site became part of the EPA Superfund program. Reclamation and closure work began in October 2007.

The Barite Hill deposit is hosted by sericitised felsic metavolcanic and metasedimentary rock of the Persimmon Fork Formation. The deposit occurs along the contact between upper and lower pyroclastic units. Mafic to intermediate post-mineralization dikes and sills cross-cut NE-trending mineralized zones. Multiple Main Pit ore zones are associated with lenses of siliceous barite rock and pyrite-quartz altered breccias, some of which are offset by normal faulting. Rainsford Pit ore zones are associated with silicified rock and chert. The Barite Hill deposit is interpreted to be the result of a Kuroko-type submarine volcanogenic base-metal sulfide system followed by epithermal precious metal deposition (Clark, 1999).

24 Other Relevant Data and Information

SRK knows of no other relevant data or information available at this time to make the technical report understandable and not misleading.

25 Interpretation and Conclusions

Geologic controls for Haile ore bodies are well understood and documented by mapping and over 4000 drill holes. Almost 70% of the holes are core holes. Mineralization is relatively robust and continuous to enable a reliable 3D geologic model to be derived. En echelon mineralized zones occur as mostly stratiform lenses within a 3.5 km long by 1 km wide area. Stratigraphic controls within the upper Persimmon Fork Formation are well constrained and crosscutting diabase dikes have been modeled. Structural controls include ENE fold axes that are locally overturned and generally plunge gently ENE. Two fault domains include ENE-striking, NW-dipping faults and NW-striking faults along diabase dike swarms. Alteration zonation is characterized by strong quartz-sericite-pyrite alteration in mineralized areas flanked by barren sericite and propylitic alteration. Saprolitization and oxidation extend to depths of 10 to 40 meters. Gold mineralization is poorly developed in saprolite due to remobilization. The Haile area is partly covered by an unmineralized southeast-thickening apron of Coastal Plain Sands.

25.1 Geology and Mineralization

Similar timing for gold mineralization and peak magmatism in the Haile and Ridgeway areas indicates that the hydrothermal systems that produced these deposits were driven by magmatism and therefore were not the product of collision, orogeny, and/or a related metamorphic event. Gold mineralization at Haile (~549 Ma, Mobley et al., 2014) slightly postdates volcanism, which precludes syngenetic and volcanogenic models and predates deformation. Gold mineralization coincides with a major tectonostratigraphic change from intermediate volcanism and tuffaceous sedimentation to basinal turbiditic sedimentation. Pressure shadows around pyrite and pyrrhotite grains, stretched hydrothermal breccia clast, and folded mineralized zones indicate that Haile mineralization is pre-tectonic. Haile is classified as a disseminated, sediment-hosted, intrusion-related gold deposit with proximal quartz-sericite-pyrite (QSP) alteration and distal sericite-chlorite alteration. (Robert et al., 2007). Haile is hosted by reduced siliciclastic rocks with less permeable volcanic caprocks, is folded and faulted, and has early K feldspar and silica-pyrite alteration with late sericite-carbonate, and is spatially associated with granitic intrusions.

Recent stratigraphic reinterpretation by OceanaGold geologists has reassigned the metasedimentary package at Haile from the Richtex Formation to the uppermost section of the Persimmon Fork Formation. This is supported by fining upward sedimentary cycles, gradational contacts, rapid facies change, tuffaceous interbeds and the occurrence of 1-3% plagioclase crystals in volcaniclastic units. Local pepperite beds tens of meters thick logged in drilling consist of alternating 0.3 cm to 2 m bands of laminated siltstone and feldspar crystal-poor felsic tuff. The conformable ENE-trending contact between the Persimmon Fork and the overlying Richtex Formation is located about 0.5 km south of Haile (figure 5). Reinterpretation of stratigraphy at Haile considerably simplifies the previous structural model with a folded volcanic-sedimentary package that is not complicated by overturning or regional thrusting.

25.1.1 Open Pit

The mine block model, geotechnical stability, pit design, phase design, dump design, production schedule and reserve estimation have been completed to a feasibility study standard. The Project confirms a positive cash flow using only Measured and Indicated Resources for the conversion of reserves using a USS1,300/oz gold price. The mine design supports the style and size of equipment selected for operations. While subject to continual improvement, the mine plan implementation will require qualified staff and the integration of all mining and related disciplines for the successful execution of the Project.

The mine operating and capital costs have been estimated from first principles and operational knowledge from current operations. The equipment is sized to meet minimum SMU requirements that support the dilution and mine recovery factors while providing bulk earthwork capability for the expected production rates.

The LoM production schedule includes provision for careful control of potentially acid generating overburden and appropriate material handling costs have been included in the mining capital and operating cost estimate.

25.1.2 Underground

OceanaGold has completed an industry standard resource definition drilling program to delineate the Horseshoe mineralization sufficient to support the current Mineral Resource estimation. The average drill spacing is approximately 25 m x 25 m within the Indicated Mineral Resource and 50 m x 50 m in the Inferred Mineral Resource. The work has been accompanied by an industry standard QA/QC program showing good quality analytical results. OceanaGold has conducted extensive core logging resulting in a high quality geologic model. The results of the drilling, sampling, analytical testing, core logging and geologic interpretation provide good support for an industry standard resource estimation. The results of the Mineral Resource estimation define an Indicated Mineral Resource of 2.71 Mt at an average Au grade of 5.68 g/t containing 0.5 Moz of gold and an additional Inferred Mineral Resource of 1.2 Mt at an average Au grade of 5.0 g/t containing 0.2 Moz of gold.

25.2 Status of Exploration, Development and Operations

Resource definition drilling at Haile by Romarco Minerals and now OceanaGold has increased the resources more than fivefold since 2007. Reserve growth resulted from 3D geologic modelling, higher gold prices, and aggressive and deeper drilling of a robust and previously underexplored mineral system. This has been recently exemplified by pre-development of the Horseshoe underground deposit and announcement of a maiden reserve (0.44 Moz) in this report. Aggressive drilling by OceanaGold continues at a rate of ~1 km of drilling per week, targeting open pit and high-grade underground mineralization proximal to the sedimentary-volcanic contact. Underground development of the Horseshoe deposit in 2019 to 2020 will facilitate access for underground drill stations along the prospective 1 km long Horseshoe-Palomino trend. Additionally, regional exploration target generation and land acquisition for Haile-like deposits has accelerated with the goal of discovering >0.5 Moz Au within 30 km of Haile.

25.3 Mining and Reserves

25.3.1 Open Pit

The Project confirms a positive cash flow using only Measured and Indicated resources for the conversion of reserves using a US$1,300/oz gold price. The mine design supports the style and size of equipment selected for operations.

The mine operating and capital costs have been estimated from first principles and operational knowledge from current mine operations. The equipment is sized to meet minimum SMU requirements that support the dilution and mine recovery factors while providing bulk earthwork capability for the expected production rates.

The LoM production schedule includes provision for careful control of potentially acid generating overburden and appropriate material handling costs have been included in the mining cost estimate.

The mine plan is based on a specific set of assumptions and are therefore the results of this Technical Report are subject to many risks including, but not limited to: commodity prices and foreign exchange assumptions (particularly relative movement of gold and oil prices), unanticipated inflation of capital or operating costs, significant changes in equipment productivities, geotechnical assumptions in pit designs, ore dilution or loss, throughput and recovery rate assumptions.

SRK knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors that might materially affect the Mineral Reserve estimate.

25.3.2 Underground

Geotechnical

A geotechnical field characterization program has been undertaken to assess the expected rock quality. This program included logging core, laboratory strength testing, in situ stress measurements and oriented core logging of jointing. The results of this program have provided adequate quantity and quality data for feasibility-level design of the underground workings.

A geotechnical assessment of the orebody shape and ground conditions has determined that long-hole open stoping mining is an appropriate mining method. Stopes have been sized to maintain stability once mucked empty. A primary/secondary extraction sequence with tight backfilling allows optimization of ore recovery while maintaining ground stability. Primary stopes will be backfilled with cemented rockfill, while secondary stopes will be backfilled with uncemented waste rock.

The design has been laid out using empirical design methods based on similar case histories. The stability of the design has been checked with 3D numerical stress-strain models of the workings which included consideration for mine-scale faulting. The modeling results confirm that stopes and access drifts are predicted to remain stable during active mining.

Mining

Longhole stoping is seen as the appropriate mining method for the deposit geometry. The large stope sizes minimize cost and grades are not overly diluted. Mine planning work considered revenue for Au and a CoG of 1.5g/t was used. Stope optimization was completed to identify economic mining areas. A 3D mine design was completed based on the stope optimization results.

The underground mine is accessed via ramp, with the ramp portal is located on an open pit bench approximately 80 m below natural surface. Two 5 m diameter raises to surface serve as intake and exhaust raises. The intake raise is equipped with an emergency egress system.

Tonnage and grades presented in the reserve include dilution and recovery and are benchmarked to other similar operations. Productivities were generated from first principles with inputs from mining contractors, blasting suppliers, and equipment vendors where appropriate. The productivities were also benchmarked to similar operations. Equipment used in this study is standard equipment used worldwide with only standard package/automation features.

A monthly/quarterly/yearly production schedule was generated using iGantt software. The schedule targeted 1,924 t/d.

25.4 Mineral Processing and Metallurgical Testing

A significant portion of equipment installed at Haile appears to have been designed conservatively enough (that is, with sufficient additional capacity) to accommodate expansion without modification.

The existing Haile facilities have sufficient additional capacity in the major equipment and site footprint to expand to a capacity of 4.0 Mtpa. This would bring the plant's major components close to their maximum capabilities. As an alternative, expansion to a greater level would require that major components of the process be rather duplicated. Doing so would cost similar amounts as the construction of the original Haile plant and would require equivalent additional space.

Identification of the whereabouts of the safety margins and other design allowances extant in the current plant will be identified during the ramp-up phase of operation. This is expected to allow operation of the plant above the current nameplate capacity and reduce the additional requirements for the proposed expansion.

No novel, experimental or unproven technologies are used for the Haile process plant or will be used in the proposed expanded plant. Many process plants of this type and size have been constructed in the past and subsequently expanded using the approach described herein.

Retrofitting of additional equipment on this scale to a plant during normal operation requires a higher degree of planning compared to construction of a new "greenfields" plant installation.

The safety issues for a ""brownfields" expansion project will be paramount to ensure all operations and contract employees are aware of the additional risks.

Detailed and well-thought-out work plans are required to minimize what will likely be inevitable operations stoppages to maintain maximum production during the expansion project construction period.

25.5 Recovery Methods

There is no effective change to the existing plant recovery method proposed for the plant expansion compared to the current circuit configuration. The processing plant is operating and is in the final stages of commissioning and entering the ramp up phase, the process selected is conventional and appropriate and in this context readily expandable to suit the accelerated processing rate.

The existing Haile facilities have sufficient additional capacity in the major equipment and site footprint to expand to a capacity of 4.0 Mtpa. This would bring the plant's major components close to their maximum capabilities. As an alternative, expansion to a greater level would require that major components of the process be rather duplicated. Doing so would cost similar amounts as the construction of the original Haile plant.

25.6 Project Infrastructure

25.6.1 Underground Support Infrastructure

The underground support infrastructure is relatively straightforward including construction of a new dry/meeting building, new underground yard, an upgrade and addition to the power distribution system, water supply system, movement of an existing shop on site to the new underground yard, and installation of a portable crusher/screen plant and CRF plant at the underground yard.

25.7 Environmental Studies and Permitting

There is a significant amount of existing background and environmental baseline data available for the Project. This data continues to be collected and reported to the regulators as part of operational controls. Additional data and environmental studies (Section 20-3) will be of technical interest to the federal and state agencies in evaluating a request to expand the current mining operation.

Permits currently held by the HGM may be kept, modified, terminated, or replaced during the expansion process.

OceanaGold will work closely with all key stakeholders to ensure that the permitting of the mine expansion meets all of the federal and state requirements.

25.8 Economic Analysis

The Project consists of a mill expansion from a starter operation operating at 6,300 t/d to a full 11,000 t/d processing capacity by January 2021. The plant is mainly fed by the OP mine which starts production at 11,000 t/d mined rate and subsequently increases production to 40,000 t/d once the mill expansion is completed in 2021. The expanded mill feed is also supplemented with ore from a five year UG 1,921 t/d high grade operation that begins production in the same year of the expanded mill capacity comes online in 2021.

The Project is expected to produce 2.854 million ounces of gold over a 16-year mine life at a rate of 182 koz Au per year with a RoM AISC of US$701/oz. The development/expansion capital cost is estimated at US$255 million that would be partially offset by a Net Preproduction Revenue credit of US$69.2 million.

Project metrics using a constant US$1,300 / oz gold price include pre-tax and after-tax NPV 5% values of US$925.3 million and US$770.5 million, respectively. Because the project is valued on a total project basis and not by an incremental analysis of the UG start up and mill expansion, an IRR value is not relevant in this analysis. In terms of sensitivity, the Project is not surprisingly most sensitive to gold grade and price following by operating costs and capital costs. The Project after-tax NPV 5% value changes US$1.50 million per US$1.00 change in the gold price.

26 Recommendations

26.1 Recommended Work Programs

26.1.1 Geology and Mineralization

Open Pit

Geologic models will be refined and expanded based largely on core drilling and pit mapping to better understand controls to mineralization which, in turn, will define new drill targets at Haile and in the region. 3D geologic domains will be defined to improve grade estimation. Structural interpretation with improved alteration zonation and geochemistry will be used to develop new drill targets within the upper Persimmon Fork Formation and in the overlying Richtex metasediments. Lithology, alteration and mineralogical data will be integrated with geotechnical, overburden storage and metallurgical evaluations to optimize and realize reserve growth. Targets will be reviewed with new geologic models and expanded pit designs.

Underground

- Additional infill drilling at 15 m x 15 m spacing could be completed in the areas of early mining to provide additional confidence in the tonnes and grade of this production to support a Measured Mineral Resource;
- A small number of longitudinal holes would serve to better define the trajectories of the swarmed cross-cutting barren dikes (these are sub-parallel to existing drilling);
- A broad development and drilling strategy to extend resources below and upwards to the SW towards the Snake Zone;
- Study grade control strategy by investigating the potential for underground, reverse circulation, grade control drilling;
- Evaluate if portable XRF testing or / other technologies can be used to further refine the geology interpretation (in tandem with in-pit studies);
- Review the entire Horseshoe geologic interpretation in early 2018, in the light of advances in geological understanding from in-pit grade control analysis, pit mapping etc.; and
- Additional sample analysis should also be conducted to refine the current NAG and PAG model.

26.1.2 Status of Exploration; Development and Operations

OceanaGold will continue to expand resources adjacent to open pit and underground reserves in the Haile district by aggressive development drilling and by regional greenfields exploration in the Carolina terrane. Systematic target generation and rationalization supported by mapping, drilling, geochemistry and geophysics is expected to yield new discoveries within 30 km of Haile during the next five years via focused exploration programs. Interpretation of geophysical data and integration with geology and geochemistry will play an important role in exploring under Coastal Plain Sands. An ‘exploration toolkit’ of diagnostic criteria for Haile-like deposits has been developed to drive exploration for potentially mineable deposits supported by a plethora of historic data and exemplary community relations.

26.1.3 Mining and Reserves

Open Pit

Equipment productivities, drill and blast efficiency and costs are all part of internal continuous improvement processes.

Implementation of the mining plan will include:

- Confirmation of the larger mining fleet selected as appropriate for the operation as more representative mining data becomes available;
- Sterilization drilling of the overburden storage areas due to the change in footprint;
- Infill drilling will likely move material from outside of a Reserve category to within. Similar changes are possible in the classification of PAG material. Additional drilling should target ultimate pit toe locations particularly where mineralization is controlled by hard boundaries or limited by inferred resources;
- Refining the mine designs to an operational level, including further optimization of the material movements and the ultimate ramp systems; and
- Further geotechnical analysis of pit design criteria as operating experience is gained and additional drilling information becomes available.

Underground

The key recommendations relating to the underground project include:

- Investigation of alternative primary ventilation intake and exhaust options through the saprolite;
- Ground support and stope sizes optimisation as more data is available; and
- The underground mobile mining fleet requires finalisation. The fleet detailed in this report lists suitable equipment based on size and capability, but there may be other suppliers of similar equipment which may be equally suitable for an underground mining operation.

26.1.4 Mineral Processing and Metallurgical Testing

The process route precious metal extraction is effectively fixed for the Haile expansion. The existing process route of generating a sulfide concentrate stream via flotation from recycle streams within the grinding circuit and from the grinding circuit product stream and subjecting this to a further grinding step before cyanidation is in place and operational.

The processing plant is operating in the final stages of commissioning and entering the ramp up phase and as such there is insufficient data available to calibrate the original design criteria. This study still draws heavily on historical metallurgical test work prepared for the current plant design.

The key areas where scope for optimization in forthcoming phases of definitive engineering study and detailed design exists are in the:

- Primary (ore) comminution circuit
- Concentrate regrind
    - specific energy requirements; and
    - optimum circuit product particle size distribution.
- Thickener unit capacity

- Oxygenation requirements for reground concentrate cyanidation

The data required to optimize the above is primarily obtained during normal plant operations with additional test work completed by external laboratories as required. The cost of the additional test work would be in the order of US$100,000.

26.1.5 Project Infrastructure

Open Pit Infrastructure

The open pit infrastructure requirements are primarily extensions of normal operations. This includes moving of facilities such as power reticulation and water diversions. The remaining requirements are waste storage facilities and main costs for these are included as sustaining capital. Additional requirements may be geotechnical investigation of the planned footprints for overburden storages.

Underground Infrastructure

The underground infrastructure should consider whether the crusher is necessary for the CRF facility and finalize the design work to an appropriate level for construction for the remaining underground infrastructure

26.1.6 Environmental Study Results

Table 26-1 is a summary of the environmental studies initiated by HGM to support the approval process.

Table 26-1: Environmental Studies

Study	Scope of Work
Air Emissions	Assess impact to air pollution loading based on additional operating conditions and new equipment on active point sources – engine exhausts, conveyor drop points, discharge stacks, ventilation shafts, dust controls, blast gasses, cement plant, etc.
Aquatic Resources – plants and animals	Create Carolina Heel Splitter mussel fish and macroinvertebrate studies to assess impact to species over LOM.
Cultural & Historical Resources	Review and assess potentially impacted cultural and historical sites from surface disturbances. Relocate any potential gravesites.
Economic Impact to Local Community	Social and economic effect on the state and local economy – effect on local businesses, wages, local resources, emergency services, and external jobs.
Geochemistry Analysis	Update OMP for PAG material placement, ARD control, and underground cut and fill practices.
Hydrology Assessment	Identification of direct and indirect impacts on local wetlands, wells, and streams.
Impacted Wetland Assessment	Assess potentially impacted wetland vegetation around Un-named Tributary, and newly acquired Jones and Gregory properties.
Reclamation Plan	Pit closure and remediation plans, surface controls, revegetation plans, stormwater control plans, surface run-off plans, timelines, and sequence for surface reclamation activities.
Surface Water Impacts	Assess impact to surface water flows – volume, dissolved oxygen, chemistry, pH and conductivity. Assess drainage patterns and develop recommendations for additional monitoring and measurement stations, if required. Create Stormwater plans for sediment ponds, location for BMP[2] devices, assessment locations, and site controls.
Terrestrial Resources – Plant Life	Perform terrestrial plant evaluations, specifically in impacted areas and areas of significant disturbance.
Terrestrial Resources – Wildlife	Perform seasonal terrestrial studies on migratory endangered wildlife, such as species of bats and raptors.
Traffic / Road Impacts	Perform a traffic study and predict road patterns and potential impacts.
Vibration Analysis	Develop vibration predictions based on underground blasting and changes in surface contours and geological morphology.
Wetland Delineation	Create surface maps with jurisdictional delineations. These will include necessary 50-ft (15.25-m) offset boundaries.
Wetland Measurement and Marking	Set the visual markers around the 50-ft offset. Measure the final size of wetland delineation.

Source: OceanaGold

[2] Best Management Practice Devices – 1) Erosion Prevention – slope surfaces, seeding, and erosion controls; and 2) Sediment Control - check dams, sediment dams, sediment ponds, and silt fencing.

26.1.7 Economic Analysis

The expanded process plant capital and operating cost estimates will be refined during the forthcoming phases of definitive engineering study and detailed design.

Optimizing the mining schedule to present a non-erratic run-of-mine ore feed blend to the plant will reduce the design envelope and allow rationalization in equipment sizing and selection in the primary (ore) and concentrate comminution circuits. This should reduce capital cost as the comminution circuits are the biggest components of the expenditure. Similarly, the operating cost is largely impacted by comminution circuit requirements by virtue of the high cost associated with energy, grinding media and mill liners in these areas.

26.2 Recommended Work Programs Costs

Table 26-1 lists the estimated costs for the recommended word described in section 26.

Table 26-1: Summary of Costs for Recommended Work

Discipline	Program Description	Cost (US$)	No Further Work is Recommended Reason:
Geology and Mineralization	External reviews	50,000	
Status of Exploration, Development and Operations	OceanaGold exploration programs and development drilling. External assay laboratories used.	100,000	
Mineral Processing and Metallurgical Testing	External laboratory testing	80,000	
Mineral Resource Estimate – Open Pit & Underground			Part of exploration programs and mine development
Mineral Reserve Estimate – Open Pit & Underground			OceanaGold Reserve reporting
Mining Methods – Open Pit & Underground	Further mine fleet and mining method analysis	150,000	
Recovery Methods			Plant constructed and operational
Project Infrastructure	Detailed design of waste storage facilities and UG requirements	60,000	
Environmental Studies and Permitting	Additional studies and legal support	500,000	
Capital and Operating Costs/Economic Analysis			OceanaGold internal
Total US$		**$940,000**	

Source: SRK, 2017

27 References

AMIRA (2002). ARD Test Handbook. Project P387A: Prediction and Kinetic Control of Acid Mine Drainage.

Ayuso, R. A., Wooden, J. L., Foley, N. K., Seal, R. R., and Sinha, A. K., 2005, U-Pb Zircon Ages and Pb Isotope Geochemistry of Gold Deposits in the Carolina Slate Belt of South Carolina: Economic Geology, v. 100, p. 225-252.

Barker, C. A., Secor, D. T., Pray, J. R., and Wright, J. E., 1998, Age and Deformation of the Longtown Metagranite, South Carolina Piedmont: A Possible Constraint on the Origin of the Carolina Terrane: Journal of Geology, v. 106, p. 713-725.

BGC Engineering Inc., 2017. "Haile Gold Mine Optimization Study - Open Pit Slope Designs", Final report issued to OceanaGold Corp. July 21, 2017

Butler, J. R., and Secor, D. T., 1991, The Central Piedmont, *in* Horton, W., and Zullo, V., eds., The Geology of the Carolinas: Knoxville, TN, University of Tennessee Press, p. 59-78.

CIM (2014). Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014.

Clark, L., (1998). Minimizing Dilution in Open Stope Mining with a Focus on Stope Design and Narrow Vein Longhole Blasting. M.Sc. thesis, Department of Mining and Mineral Process Engineering, The University of British Columbia, 316 p.

Clark, L., and R. Pakalnis (1997). An Empirical Design Approach for Estimating Unplanned Dilution from Open Stope Hangingwalls and Footwalls. Presentation at 99th Canadian Institute of Mining annual conference, Vancouver, B.C.

Clark, S., 1999, Geologic Maps and Block Diagrams of the Barite Hill Gold-Silver Deposit and Vicinity, South Carolina and Georgia, U.S. Geological Survey Open-File Report 99-148A

Dooley, R. E., and Smith, W. A., 1982, Age and magnetism of diabase dykes and tilting of the Piedmont: Tectonophysics, v. 90, p. 283-307.

Daniels, D., 2005 South Carolina Aeromagnetic and Gravity Maps and Data, USGS Open-File Report 2005-1022

Feasibility Level Pit Slope Evaluation, Golder Associates, October 2010.Environmental Impact Statement, Haile Gold Mine Project, July 2014

Feiss, P. G., Vance, R. K., and Weslowki, D. J., 1993, Volcanic rock hosted gold and base metal mineralization associated with Neoproterozoic-Early Paleozoic back-arc extension in the Carolina terrane, southern Appalachian Piedmont: Geology, v. 21, p. 439-442.

Foley, N. K., Ayuso, R. A., and Seal, R. R., 2001, Remnant colloform pyrite at the Haile gold deposit, South Carolina: A textural key to genesis: Economic Geology, v. 96, p. 891-902.

First Quarter 2017 Results April 27, 2017 www.oceanagold.com

Fishes in Camp Branch, Fred C. Rohde, August 2010.

Fullagar, P. D., and Butler, J. R., 1979, 325 to 265 Ma-old granitic plutons in the Piedmont of the southeastern Appalachians: *American Journal of Science,* v. 279, p. 161-185

Gillon, K.A., Spence, W.H., Duckett, R.P., and Benson, C.J., 1995, Geology of the Ridgeway gold deposits, Ridgeway, South Carolina: Society of Economic Geologists Guidebook Series, v. 24, p. 53–94.

Gillon, K.A., Mitchell, T.L., Dinkowitz, S.R., Barnett, R.L., 1998, The Ridgeway gold deposits—A window to the evolution of a Neoproterozoic intra-arc basin in the Carolina terrane, South Carolina: South Carolina Geology, 40, p. 29–70.

Gillon, K., 2000, Final observations from the Ridgeway North Pit, and their bearing on a tectonic development model for the Ridgeway gold deposits, Carolina Terrane, SC, GSA abstract, March, 2000, Vol. 32, Issue 2, pp. 21

Gillon, K., 2012, The Ridgeway gold deposits, South Carolina, Some old and new ideas regarding possible origins for these fossil hyrdrothermal gold deposits, Geological Society of America *Abstracts with Programs.* Vol. 44, No. 7, p.198

Golder (2010). Haile Gold Mine, Lancaster County, South Carolina, Report on Feasibility Level Pit Slope Evaluation, Project No. 093-91767.300, October 2010.

Golder (2012). Preliminary Geotechnical Investigation, Horseshoe Deposit, Haile Gold Mine, Lancaster County, South Carolina, Appendix A: Rock Core Logging Data, Project No. 11390150, January 5, 2012.

Haile Gold Mine Addendum to Baseline Geochemistry Report, Schafer Limited LLC, November 2010.

Haile Gold Mine Baseline Hydrologic Characterization Report, Schlumberger Water Services, November 2010.

Haile Gold Mine Depressurization and Dewatering Feasibility Study, Schlumberger Water Services, December 2010.

Haile Gold Mine Project NI 43-101 Technical Report Feasibility Study Lancaster County, South Carolina. M3 Engineering & Technology Corporation, 10 February 2011.

Haile Gold Mine Project Resource Estimate NI 43-101 Technical Report Lancaster County, South Carolina. M3 Engineering & Technology Corporation, March 13, 2012.

Haile Power Supply Recommendations for Feasibility Study, Haile Gold Mine, Inc., December 2010.

Hardy, L. S., 1989, Hydrothermal potassium feldspar at the Haile gold mine, South Carolina: Economic Geology, v. 84, p. 2307-2310.

Hatcher, R. D., Bream, B. R., and Merschat, A. J., 2007, Tectonic map of the southern and central Appalachians: A tale of three orogens and complete Wilson cycle, *in* Hatcher, R. D., Carbon, M. P., McBride, J. H., and Martínes Catalán, J. R., eds., Framework of Continental Crust: Geological Society of American Memoir, v. 200, (Eds.), 2007. 4D Framework of Continental Crust. Geological Society of America, Memoir, vol. 200. 641 pp., p. 595-632.

Hayward, N., 1992, Controls on syntectonic replacement mineralization in parasitic antiforms, Haile gold mine, Carolina Slate Belt, U. S. A.: Economic Geology, v. 87, p. 91-112.

Hibbard, J. P., Shell, G., Bradley, P., Samson, S., and Wortman, G., 1998, The Hyco shear zone in North Carolina and southern Virginia: implications for the Piedmont zone-Carolina zone boundary in the southern Appalachians: American Journal of Science, v. 298, p. 85 - 107.

Hatcher, Robert D, Jr. Carlson, Marvin P., McBride, John H, Catalan, Jose R. Martinez. (2007) “4D Framework of Continental Crust (Memoirs (Geological Society of America))”. Geological Society of America

Hibbard, J.P., van Staal, C.R., and Millar, B.V., 2007, Links between Carolinia, Avalonia, and Ganderia in the Appalachian per-Gondwanan Realm; in Sears, J., Harms, T., and Evenchick, C., eds., From Whence the Mountains?: Inquiries into the Evolution of Orogenic Systems: A Volume in Honor of Raymond A. Price: GSA Special Paper 433, p. 291-311.

Hibbard, J.P., van Staal, C.R., and Rankin, D.W., 2010, Comparative analysis of the geological evolution of the northern and southern Appalachian orogen: Late Ordivician-Permian: in Tollo, R.P., Bartholomew, M.J., Hibbard, J.P., and Karabinos, P.M., eds., From Rodinia to Pangea: The Lithotectonic Record of the Appalachian Region: Geological Society of America Memoir 206, p. 51-69.

Intermediate List of Vascular Plant Species Noted During Surveys of the Haile Gold Mine Properties May, 2008 to August, 2010, Ecological Services and Consulting, September 2010.

Larson, A., and Worthington, J.,1989, Geophysical mapping at the Haile mine, report for Piedmont Mining Company, January 1989, 22p.

Lawrence, R.W., Poling, G.W. and Marchant, P.B., (1989). Investigation of predictive techniques for acid mine drainage. Report on DSS Contract No. 23440-7-9178/01-SQ, Energy Mines and Resources, Canada, MEND Report 1.16.1(a).M3 (2014). Haile Gold Mine Project: NI 43-101 Technical Report Project Update.

M3, 2015. Haile Gold Mine Project, NI 43-101 Technical Report Project Update, Lancaster County, South Carolina. October 13, 2015.

Maddry, J.W., and Kilbey, T.R., 1995, Geology of the Haile gold mine, selected mineral deposits of the Gulf Coast and southeastern United States: Part II. Gold deposits of the Carolina slate belt: Society of Economic Geologists Guidebook Series, v. 24, p. 147−172.

Maher, H.; Sacks, PE.; Secor, Donald T. (1991). “The eastern Piedmont in South Carolina”. Geology of the Carolinas: Carolina Geological Society 50th Anniversery. University of Tennesee.

Mathews, K.E., Hoek, E., Wyllie, D.C. and Stewart, S.B.V. (1981). Prediction of stable excavations for mining at depth below 1000 meters in hard rock. CANMET Report DSS Serial No. OSQ80-00081, DSS File No. 17SQ.23440-0-9020. Ottawa: Dept. Energy, Mines and Resources.

McCauley, C. K., and Butler, J. B.,1966, Gold resources of South Carolina, Bulletin 32, State Development Board, South Carolina, 76 p.

Memorandum on the Report on Survey for mollusks on Camp Branch of the Little Lynches River, Lancaster County, South Carolina,Eugene P. Keferl, September, 2010.

Metso Minerals Industries, Inc., (Metso), York, Pennsylvania, December 7, 2009, Test Plant Report No. 20000134-135.

Mine Water Treatment Plant Preliminary Design Report, Schlumberger Water Services, December 2010.

Mobley, R.M., Yogodzinski, G.M., Creaser, R.A., and Berry, J.M., 2014, Geologic History and Timing of Mineralization at the Haile Gold Mine, South Carolina, Economic Geology, v. 109, p. 1863-1881.

Moye, R., 2012, Evidence of synkinematic magmatism and mineralization in the Neoproterozoic sediment-hosted Kennecott Ridgeway Au-Ag-Mo deposit, South Carolina, Geological Society of America *Abstracts with Programs.* Vol. 44, No. 7, p.198

Newton, Edmund; Gregg, D.B.; Mosier, McHenry. (1940). “Operations at the Haile gold mine, Kershaw, SC” Information Circular: Volume 7111, U.S Department of the interior: Bureau of Mines.

Nystrom, P. G., Jr., Willoughby, R. H., and Price, L. K., 1991, Cretaceous and Tertiary stratigraphy of the upper Coastal Plain, South Carolina, *in* Horton, J. W., Jr., and Zullo, V. A. eds., The Geology of the Carolinas: Knoxville, Tennessee, The University of Tennessee Press, p. 221-240

Pardee, J.T.; Park Jr., C.T. (1948) “Gold deposits of the southern Piedmont” USGS Numbered Series, Professional Paper 213, U.S Geological Survey.

Potvin Y. & J. Hadjigeorgiou (2001). The Stability Graph Method for Open Stope Design. Chapter 60 in Underground Mining Methods. Hustrulid & Bullock Eds. Society of Mining Engineers, pp. 513-520

Price, (2009). Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials.

Price, W.A. (2009). Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials.Price, W.A. (1997). Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Minesites in British Columbia—DRAFT. British Columbia Ministry of Employment and Investment. Smithers, BC. April 1997.

Price, W.A., (1997). Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Minesites in British Columbia—DRAFT. British Columbia Ministry of Employment and Investment. Smithers, BC. April 1997.

Reclamation Plan Overview, HGM, December 2010.

Record of Decision, Haile Gold Mine Project, October 27, 2014

Robert, F., Brommecker, R., Bourne, B., Dobak, P., McEwan, C., Rowe, R., and Zhou, X., 2007 Models and exploration methods for major gold deposit types, Ore deposits and exploration technology, in Proceedings of exploration 07; Fifth decennial international conference on mineral exploration, ed. B. Milkereit, p. 691-711Romarco_Minerals, 2010, Press Release - Romarco Announces In-Shell Resource Estimate, November 7, 2010, http://www.romarco.com/news/2010/index.php?&content_id=332

Romarco Minerals, Inc. Flash Flotation, Cyanide Destruction & Leaching Of Concentrate and Tailing for Haile Composites, Resource Development Inc., (RDi), Wheat Ridge, Colorado, May 27, 2010.

Romarco Minerals, Inc. Haile Gold Project, Metallurgical Report, Resource Development Inc., (RDi), Wheat Ridge, Colorado, September 16, 2009.

Romarco Minerals, Inc. Metallurgical Testing Of Ledbetter Extension Samples, Resource Development Inc., (RDi), Wheat Ridge, Colorado, March 31, 2010.

Romarco Minerals, Inc. Optimization Of Leaching Of Flotation Concentrate, Resource Development Inc., (RDi), Wheat Ridge, Colorado, September 27, 2010.

Romarco Minerals, Inc. Work Index Data For Haile Composite Sample, Resource Development Inc., (RDi), Wheat Ridge, Colorado, March 31, 2010.

Schafer Limited LLC (2015). Revised Haile Gold Mine Overburden Management Plan. September 2015. 25 pp.

Schafer Limited LLC, (2015). Revised Haile Gold Mine Overburden Management Plan

Schafer Limited LLC. (2010a). Haile Gold Mine Baseline Geochemistry Report. October, 2008, Revised November 2010. 78 pp.

Schafer Limited LLC. (2010b). Haile Gold Mine, Addendum to Baseline Geochemistry Report. November, 2010. 63 pp.

Schafer Limited LLC. (2012). Haile Gold Mine, Second Addendum to Baseline Geochemistry Report. March, 2012. 60 pp.

Schafer Limited LLC. (2013). Haile Gold Mine, Third Addendum to Baseline Geochemistry Report. January, 2013. 69 pp.

Scheetz, J.W., J.M. Stonehouse, and M.R. Zwaschka. 1991, "Geology of the Brewer Gold Mine in South Carolina." *Mining Engineering*, pp. 38-42

Search for Federal Threatened and Endangered Species and State Species of Concern on the Duckwood Tract, Needham Environmental Inc., October 2010.

Secor, Donald T. Jr.; Snoke, Arthur W.; (1986). “Character of the Alleghanian orogeny in the Southern Appalachians: Part II. Geochronological cosntraints on the tectonothermal evolution of the eastern Peidmont in South Carolina.” GSA Bulletin, Volume 97, Number 11.

Secor, D. T., Jr., and Wagener, H. D., 1968, Stratigraphy, structure and petrology of the Piedmont in Central South Carolina: Columbia, South Carolina, South Carolina Geological Survey, Geologic Notes, v. 12, p. 67-84.

Secor, D.T., and Snoke, A.W., 1978, Stratigraphy, structure, and plutonism in the Central South Carolina Piedmont, in Snoke, A.W., ed., Geological Investigations of the Eastern Piedmont, Southern Appalachians: Carolina Geological Society Guidebook for 1978 Annual Meeting, p. 43−63.

Secor, D. T., Samson, S. L., Snoke, A. W., and Palmer, A. R., 1983, Confirmation of the Carolina Slate Belt as an Exotic Terrane: Science, v. 221, p. 649-651.

Sobek, A., Schuller, Freeman, W.J. and Smith, R. (1978). Field and Laboratory Methods Applicable to Overburdens and Minesoil, West Virginia Univ., Morgantown College of Agriculture and Forestry. EPA report no. EPA-600/2-78-054 p.47-50.O’Kane Consultants, 2015. Project Queenstown: AMD Review and Waste Block Modeling. March 9, 2015. 41 pps.

Speer, W. E., and Maddry, J. W., 1993, Geology and recent discoveries at the Haile gold mine, Lancaster County, South Carolina: South Carolina Geology, v. 35, p. 9-26.

Spence, W. H., Maddry, J. W., Worthington, J. E., Jones, E. M., and Kiff, I. T., 1980, Origin of the gold mineralization at the Haile gold mine, Lancaster County, South Carolina: Mining Engineering, v. 32, p. 70-73.

SRK Consulting (2016a). NI 43-101 Technical Report: Preliminary Economic Assessment, Haile Gold Mine, Lancaster County, South Carolina. September 8, 2016, 126 pp.

SRK Consulting (2016b). Sample Selection and Analysis for Geochemical Testing, Haile Underground Scoping Study.

SRK Consulting (2017). Geochemical Summary Report, Feasibility Study, Haile Gold Mine, Lancaster County, South Carolina. Report prepared for OceanaGold Corporation. June 2, 2017, 68 pp.

Stein, H. J., Markley, R. J., Morgan, J. W., Hannah, J. L., Zak, K., and Sundblad, K., 1997, Re-Os dating of sheer-hosted Au deposits using molybdenite, *in* Papunen, H., ed., Mineral Deposits: Research and Explorations: Where Do They Meet? Proceedings of the Fourth Biennial SGA Meeting, Turku, Finland, 11-13 August 1997, p. 313-317.

Tailing and Process Water Management Bankable Feasibility Study, AMEC Earth and Environmental, November 19, 2010.

Tomkinson, M. J., 1988, Gold mineralization in phyllonites at the Haile Mine, South Carolina: Economic Geology, v. 83, p. 1392-1400.

Weis, T., 2016, Extended Haile geophysical interpretation, report for OceanaGold, December 2016, 143 p.

Worthington, J. E., 1993, The Carolina slate belt and its gold deposits: Reflections after a quarter century: South Carolina Geology, v. 35, p. 1-8.

Worthington, J. E., and Kiff, I.T., , 1970, A suggested volcanogenic origin for certain gold deposits in the slate belt of the Carolina piedmont:: Economic Geology, v. 65, p. 529-537.

28 Glossary

The Mineral Resources and Mineral Reserves have been classified according to CIM (CIM, 2014). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

28.1 Mineral Resources

A **Mineral Resource** is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

28.2 Mineral Reserves

A **Mineral Reserve** is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.

A **Probable Mineral Reserve** is the economically mineable part of an Indicated Mineral Resource, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.

A **Proven Mineral Reserve** is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

28.3 Definition of Terms

The following general mining terms may be used in this report.

Table 28-1: Definition of Terms

Term	Definition
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All other expenditures not classified as operating costs.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade (CoG)	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution	Waste, which is unavoidably mined with ore.
Dip	Angle of inclination of a geological feature/rock from the horizontal.
Fault	The surface of a fracture along which movement has occurred.
Footwall	The underlying side of an orebody or stope.
Gangue	Non-valuable components of the ore.
Grade	The measure of concentration of gold within mineralized rock.
Hangingwall	The overlying side of an orebody or slope.
Haulage	A horizontal underground excavation which is used to transport mined ore.
Hydrocyclone	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological	Geological description pertaining to different rock types.
LoM Plans	Life-of-Mine plans.
LRP	Long Range Plan.
Material Properties	Mine properties.
Milling	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Mining Assets	The Material Properties and Significant Exploration Properties.
Ongoing Capital	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore Reserve	See Mineral Reserve.

Term	Definition
Pillar	Rock left behind to help support the excavations in an underground mine.
RoM	Run-of-Mine.
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste.
Sill	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope	Underground void created by mining.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide	A sulfur bearing mineral.
Tailings	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures including those of an operating and capital nature.
Variogram	A statistical representation of the characteristics (usually grade).

28.4 Abbreviations

The following abbreviations may be used in this report.

Table 28-2: Abbreviations

Abbreviation	Unit or Term
%	percent
~	approximately
°	degree
°C	temperature in degrees Celsius
°F	temperature in degrees Fahrenheit
2D	two-dimensional
3D	three-dimensional
AA	atomic absorption
AHK	AHK Geochem
AISC	All-In Sustaining Cost
ALS	ALS Limited
AP	acid generating potential
Ar	argon
ARD	acid rock drainage
ASTM	American Society for Testing and Materials
Au	gold
bhp	brake horsepower
BLM	United States Department of the Interior Bureau of Land Management
BM	Bond ball mill
Breccia	brecciated rocks
$CaCO_3$	Calcium carbonate
Cat	Caterpillar
cfm	cubic feet per minute
CIL	Carbon-In-Leach
CIT	Corporate Income Tax
cm^3	cubic centimeter
CoG	cut-off grade
CPLEX	IBM ILOG CPLEX Optimizer

Abbreviation	Unit or Term
CPS	costal plain sands
CRF	cemented rock fill
CRM	Certified Reference Materials
CV	coefficient of variation
Cyprus	Cyprus Exploration Company
DB	intrusive dykes
DDH	diamond drilling
DHEC	Department of Health and Environmental Control
DSS	direct shear strength
EIS	Environmental Impact Statement
ELOS	equivalent linear overbreak/slough
EPCM	Engineering, Procurement and Construction Management
Excel	Microsoft
FA	fire assay
FDD	Fresh dikes
FF	Fracture frequency
FF/m	frequency fracture per meter
Fill	back-fill from historical mining
FMS	Fresh metasediment
FMV	Fresh metavolcanic
FoS	factor of safety
FS	feasibility study
ft	foot (feet)
g	gram
g/t	grams per tonne
g/t	grams per tonne
geology data	GL
GPa	gigapascal
gpm	gallons per minute
ha	hectares
Haile	Haile Gold Mine
HDPE	height density polyethylene
HGM	Haile Gold Mine, Inc.
HMC	Haile Mining Compnay
HMV	
hp	horsepower
hr	hour
ICP-MS	inductively coupled plasma mass spectrometry
IDW^2	Inverse Distance Weighting Squared
IMC	Independent Mining Consultants
in	inch
IP	Induced polarization
IRR	initial rate of return
IRS	intact rock strength
ISO/IEC	International Electrotechnical Commission
ISO-9001	International Organization for Standardization
ISRM	International Society of Rock Mechanics
Ja	joint alteration
Jn	joint number
JPAG	Johnny’s PAG
Jr	joint roughness
kg	kilogram
km	kilometer
KML	Kershaw Mineral Lab
KML	Kershaw Mineral Lab
kN	kilonewton
kN/m^3	kilonewton per cubic meter
koz	thousand troy ounce
KS	Komologorov-Smirnoff

Abbreviation	Unit or Term
kt	thousand tonnes
kV	kilovolt
kW	kilowatt
L	liter
lb	pound
Leapfrog	ARANZ Leapfrog® Geo software
LECO	LECO Corporation
LHD	long-haul-dump
LoM	life-of-mine
m	meter
m/d	meter per day
m^3	cubic meter
Ma	mega-annum
Ml	laminated metasediments
MIBC	methyl isobutyl carbinol
min	minute
ML	metal leaching
mm	millimeter
MOA	Memorandum of Agreement
MPa	megapascal
MS	metasediments
Ms	silicified metasediments
Mst	million short tons
Mtpa	million tonnes per annum
MV	metavolcanics
MW	million watts
N’	stability number
N	north
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
NN	nearest neighbor
NNP	net neutralization potential
NOL	Net Operating Losses
NP	acid neutralization potential
NPDES	National Pollutant Discharge Elimination System
NPR	neutralization potential ratio
NPV	net present value
OC	oriented core data
OceanaGold	OceanaGold Corporation
OK	Ordinary Kriging
OMP	Overburden Management Plan
OP	open pit
OSA	overburden storage area
oz	ounce
PAG	potential acid generating
PAX	potassium amyl xanthate
PEA	preliminary economic assessment
Piedmont	Piedmont Mining Company
PLT	point load test
PMP	Probable Maximum Precipitation
Project	Haile Gold Mine
POF	probability of failure
ppb	parts per billion
ppm	parts per million
QA	Quality Assurance
QC	Quality Control
QP	Qualified Persons
QSP	quartz-sericite-pyrite
RC	reverse circulation

Abbreviation	**Unit or Term**
rd	rounds
RDi	Resource Development Inc.
RM	rock mass
RMI	Romarco Minerals, Inc.
RMR	Rock Mass Rating
RoM	run-of-mine
RQD	rock quality designation
S	sulfur
SAG	Semi-Autogenous Grinding
SAP	sampling and analysis plan
SAP	Saprolite
SCDHEC	South Carolina Department of Health and Environmental Control
sec	second
SMC	Sag Mill Comminution
SMU	selective mining unit
SO_2	sulfur dioxide
SRF	stress reduction factor
SRK	SRK Consulting (U.S.), Inc.
st	short ton (2,000 pounds)
S_T	total sulfur
st/d	short tons per day
STD	standard deviation
t/d	metric tonnes per day
TCC	total cash costs
TCR	total core recovery
TCS	triaxial compressive strength
TIC	total inorganic carbon
TSF	tailings storage facility
TV	televiewer data
UCS	uniaxial compressive strength
UG	underground
U-Pb	Uranium–lead
US$	United States Dollar
USA	United States of America
USFS	United States Forest Service
V	volts
VFD	variable frequency drive
W	watt
W:O	waste:ore
WAD	weak acid dissociable
WDD	weathered dikes
Wi	work indicies
WMS	Weathered metasediment
WMV	weathered metavolcanic
y	year
µm	microns

Appendices

Appendix A: Certificates of Qualified Persons

CERTIFICATE OF QUALIFIED PERSON

I, David Read Carr, MAusIMM CP (Met) do hereby certify that:

1. I am Chief Metallurgist of Oceanagold Corpration, 22 Maclaggan Street, Dunedin, New Zealand.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Feasibility Study Haile Gold Mine Lancaster County, South Carolina" with an Effective Date of January 1, 2017 (the "Technical Report").
3. I graduated with a degree in Metallurgical Engineering from the University of South Australia in 1993. I am a Member and Chartered Professional of the Australasian Institute of Mining and Metallurgy. I have worked as a Metallurgist for a total of 24 years since my graduation from university. My relevant experience includes base metal flotation, flotation and leaching of gold ores, pressure oxidation of refractory sulphide ores, ultrafine grinding, process plant design and plant commissioning.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I last visited the Haile Gold Mine property from March 28th 2017 until July 18th 2017.
6. I am responsible for the preparation of Sections 1.4, 1.8, 13,17, 21, 25.4, 25.5 and 26.1.4 of the Technical Report.
7. I am not independent of the issuer applying all of the tests in section 1.5 of NI 43-101 because I am an employee of OceanaGold (New Zealand) Limited.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is the preparation of the "NI 43-101 Technical Report Preliminary Economic Assessment Haile Gold Mine Lancaster Count, South Carolina" dated July 20, 2016.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

David Read Carr, MAusIMM CP (Met)

Oceana Gold (New Zealand) Ltd
(Incorporated in New Zealand
NZBN 9429 0377 53023)

22 Maclaggan Street
Dunedin, 9016
New Zealand

PO Box 5442
Dunedin, 9058
New Zealand

Telephone: 64 3 479 2922
Facsimile: 64 3 477 6708
Website: www.oceanagold.com

CERTIFICATE OF QUALIFIED PERSON

I, Bruce H. van Brunt, MSc., B.A., Fellow AusIMM do hereby certify that:

1. I am Technical Services Manager of OceanaGold, 6911 Snowy Owl Road, Kershaw, SC.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Feasibility Study Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of 1 January 2017 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from University of Nevada-Reno in 1990. In addition, I have obtained a Geology BA from the Unversity of Colorado in 1983. I am a Fellow of the AusIMM. I have worked as a Mining Engineer for a total of 32 years since my graduation from university. My relevant experience includes more than one year of continuous employment at the Haile Gold Mine in the capacity of Technical Services Manager.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have been employed at the Haile Mine property since the 23rd of June 2016
6. I am responsible for the preparation of Sections 4.1, 4.2, 5.1, 5.2, 5.3, 5.4, 5.5, 14.1.1, 14.1.2, 14.1.3, 14.1.4, 14.1.5, 14.1.6, 14.1.7, 14.1.8, 14.1.9, 14.1.10, 14.1.11, 14.1.12, 14.1.13, 14.1.14, 14.1.15, 14.1.16, 16.1.1, 16.1.3, 16.1.5, 16.1.7, 18.5, 18.6, 18.7, 18.8, 18.9, 20, 20.1, 20.2, 20.3, 21.1, 21.1.1, 23, 23.1, 23.2, 23.3, co-author 14, 14.3, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am not independent of the issuer applying all of the tests in section 1.5 of NI 43-101. I am currently employed by OceanaGold.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is employment as the Technical Services Manager.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

Bruce H. van Brunt

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CERTIFICATE OF QUALIFIED PERSON

I, John Jory, do hereby certify that:

1. I am Director of Exploration, Haile Gold Mine of OceanaGold, 6911 Snowy Owl Road, Kershaw, SC 29067, USA.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Feasibility Study, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of January 1, 2017 (the "Technical Report").
3. I graduated with a degree in B.Sc. Geology from University of Southampton, U.K., in 1982. In addition, I have obtained an M.Sc. Mineral Exploration degree in 1983 from the Royal School of Mines, Imperial College, London, U.K.. I am a Certified Professional Geologist #11902 of the American Institute of Professional Geologists. I have worked as a Geologist for a total of 30 years since my graduation from university. My relevant experience includes grade control, pit mapping, geotechnical and metallurgical evaluations, core and RC chip logging, data validation and analysis, 3D geologic modeling, basic geostatistics, geochemical and geophysical surveys and data evaluation, technical report writing, and management of geological and drilling teams.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have worked as an employee of OceanaGold at the Haile gold mine property since August 24, 2016.
6. I am responsible for the preparation of Sections 6, 7.1, 7.2, 7.2.1, 7.2.2, 7.2.3, 8, 8.1, 8.2, 9.1, 9.2, 9.3, 9.3.1, 9.3.2, 10.1, 10.2, 10.3, 10.4, 11.1, 11.1.1, 11.1.2, 11.2, 11.3, 11.4, 11.4.1, 11.4.2, 11.4.3, 11.4.4, 11.5, 12, 12.1, 12.1.1, 12.1.2, 12.2, 12.2.1, 12.3, 12.3.1, 12.3.2, 12.3.3, 12.4, 12.4.1, 12.4.2, 12.4.3, 12.5, 12.5.1, 12.5.2, 12.5.3, 12.6, 12.6.1, 12.6.2, 12.6.3, 12.6.4, 12.7, 12.7.1, 12.7.2, 12.7.3, 12.7.4, 12.7.5, 12.7.6 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report."
7. I am not independent of the issuer applying all of the tests in section 1.5 of NI 43-101. My team provides the data and geologic interpretation that guides the reserve estimates in the Technical Report.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is I have worked as an employee of OceanaGold at the Haile gold mine property since August 24, 2016.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th day of August, 2017

John Jory

CPG # 11902, AIPG

[Seal or Stamp]

John Jory

238 Angas Street
Adelaide
SA 5000
Telephone: +61 8 8232 4888
Fax: +61 7 3828 6999

9 August 2017

RE: **Certificate of qualified person**

I, Paul Howe, BSc. MAppSc. MAIG, do hereby certify that:

1. I am a Managing Principal and Principal Hydrogeologist of CDM Smith Australia Pty Ltd, Level 2, 238 Angas Street, Adelaide, South Australia.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Feasibility Study
3. Haile Gold Mine Lancaster County, South Carolina" with an Effective Date of January 1, 2017 (the "Technical Report").
4. I graduated with a degree in Biological and Earth Sciences from Monash University in 1981. In addition, I have obtained a Master of Applied Science Hydrogeology and Groundwater Management, 1993, University of New South Wales. I am a Member of the Australian Institute of Geoscientists. I have worked as a Hydrogeologist for a total of 26 years since my graduation from university. My relevant experience includes undertaking regional and local scale hydrogeological characterisations and groundwater impact assessments for mining and energy sector clients, and government agencies, analytical and numerical groundwater modeling, mine water management studies as well as playing a lead role in a number of guidance documents such as the National Framework for Assessing the Effects of Mining on Groundwater and Connected Systems (for Australia's National Water Commission).
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I visited the Haile Gold Mine property between November 27 and December 6 on , 2016 for 7 days.
7. I am responsible for Sections 16.1.8, 16.2.3, 18.4, co-author 16.1.2, and portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report..
8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
9. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is assisting project acquisition due diligence, optimisation of mine pit dewatering and depressurisation, and the underground scoping study.
10. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
11. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

Paul Howe

SRK Consulting (U.S.), Inc.
Suite 600
1125 Seventeenth Street
Denver, CO 80202

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CERTIFICATE OF QUALIFIED PERSON

I, Joanna Poeck, BEng Mining, SME-RM, MMSAQP, do hereby certify that:

1. I am a Senior Mining Engineer of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Feasibility Study, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of January 1, 2017 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Colorado School of Mines in 2003. I am a Registered Member of the Society of Mining, Metallurgy & Exploration Geology. I am a QP member of the Mining & Metallurgical Society of America. I have worked as a Mining Engineer for a total of 14 years since my graduation from university. My relevant experience includes open pit and underground design, mine scheduling, pit optimization and truck productivity analysis.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile Gold Mine property on February 6, 2017 for 2 days.
6. I am responsible for the preparation of Sections 2.1, 2.2, 2.3, 2.4, 2.5, 2.6, 3, 15, 15.2, 15.2.1, 15.2.2, 15.2.3, 16, 16.2, 16.2.1, 16.2.5, 16.2.6, 16.2.8, 24, 27, 28, 28.1, 28.2, 28.3, 28.4, co-authored 15.3, 16.3, and portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is I served as QP in a prior Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

"Signed" "Sealed"

Joanna Poeck, BEng Mining, SME-RM [4131289RM], MMSAQP[01387QP]
Senior Consultant (Mining Engineer)

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CERTIFICATE OF QUALIFIED PERSON

I, Jeff Osborn, BEng Mining, MMSAQP do hereby certify that:

1. I am a Principal Consultant (Mining Engineer) of SRK Consulting (U.S.), Inc., 1125 Seventeenth, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Feasibility Study, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of January 1, 2017 (the "Technical Report").
3. I graduated with a Bachelor of Science Mining Engineering degree from the Colorado School of Mines in 1986. I am a Qualified Professional (QP) Member of the Mining and Metallurgical Society of America. I have worked as a Mining Engineer for a total of 29 years since my graduation from university. My relevant experience includes responsibilities in operations, maintenance, engineering, management, and construction activities.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile Gold Mine property on February 6, 2017 for 2 days.
6. I am responsible for the preparation of Sections 16.2.7, 16.2.9, 16.2.11, 18.7.1, and portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
9. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

"Signed" "Sealed"

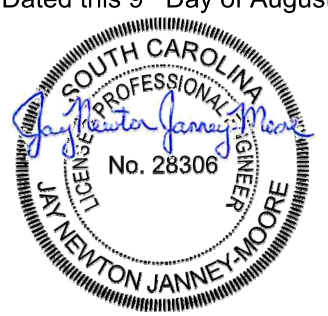
Jeff Osborn, BEng Mining, MMSAQP [01458QP]

CERTIFICATE OF QUALIFIED PERSON

I, Jay Newton Janney-Moore, PE do hereby certify that:

1. I am employed as an engineer at NewFields Mining & Technical Services LLC ("NewFields"), 9400 Station Street, Suite 300, Lone Tree, Colorado 80124, USA.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Feasibility Study Haile Gold Mine Lancaster County, South Carolina" with an Effective Date of 1st of January 2017 (the "Technical Report").
3. I graduated with a degree in Bachelor of Science in Civil Engineering from University of Colorado Denver in 1998. I am a registered professional engineer in the State of South Carolina (No. 28306) and in the State of Colorado (No. 37571). I have worked as an engineer a total of 19 years since my graduation from university. My relevant includes designing heap leach pads, tailings storage facilities, surface water diversions, and other supporting infrastructure.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile Gold Mine property on 15th of March 2017 for 3 days
6. I am responsible for the preparation of Sections 18.1, 18.2 and 18.3 of the Technical Report and portions of Sections 1, 25, and 26 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is the engineer of record for the Potentially Acid Overburden Storage Area and Contact Water Ponds and Duckwood Tailings Storage Facility.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.



Jay Newton Janney-Moore

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CERTIFICATE OF QUALIFIED PERSON

I, John Tinucci, Ph.D., P.E., ISRM, do hereby certify that:

1. I am a Principal Geotechnical Mining Engineer of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report Feasibility Study Haile Gold Mine Lancaster County, South Carolina” with an Effective Date of January 1, 2017 (the “Technical Report”).
3. I graduated with a degree in B.S. in Civil Engineering from Colorado State University, in 1980. In addition, I have obtained a M.S. in Geotechnical Engineering from University of California, Berkeley, in 1983 and I have obtained a Ph.D. in Geotechnical Engineering, Rock Mechanics from the University of California, Berkeley in 1985. I am member of the American Rock Mechanics Association, a member of the International Society of Rock Mechanics, a member of the ASCE GeoInstitute, and a Registered Member of the Society for Mining, Metallurgy & Exploration. I have worked as a Mining and Geotechnical Engineer for a total of 37 years since my graduation from university. My relevant experience includes 34 years of professional experience. I have 15 years managerial experience leading project teams, managing P&L operations for 120 staff, and directed own company of 8 staff for 8 years. I have technical experience in mine design, prefeasibility studies, feasibility studies, geomechanical assessments, rock mass characterization, project management, numerical analyses, underground mine stability, subsidence, tunneling, ground support, slope design and stabilization, excavation remediation, induced seismicity and dynamic ground motion. My industry commodities experience includes salt, potash, coal, platinum/palladium, iron, molybdenum, gold, silver, zinc, diamonds, and copper. My mine design experience includes open pit, room and pillar, (single and multi-level), conventional drill-and-blast and mechanized cutting, longwall, steep narrow vein, cut and fill, block caving, sublevel caving and cut and fill longhole stoping and paste backfilling.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I visited the Haile Gold Mine property on November 29, 2016 for 3 days.
6. I am responsible for the preparation of Section 16.2.2 and portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

“Signed” “Sealed”

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CERTIFICATE OF AUTHOR

I, Bret C. Swanson, B.Eng. (Mining), MAusIMM, MMSAQP do hereby certify that:

1. I am Principal Mining Engineer of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled ""NI 43-101 Technical Report Feasibility Study Haile Gold Mine Lancaster County, South Carolina" with an Effective Date of January 1, 2017 (the "Technical Report").
3. I graduated with a degree in Bachelor of Engineering in Mining Engineering from the University of Wollongong in 1997. I am a current member of the Mining & Metallurgical Society of America #01418QP. I have worked as a Mining Engineer for a total of 20 years since my graduation from university. My relevant experience includes contributions to numerous feasibility, pre-feasibility, preliminary assessment and competent person reports while employed with SRK, Denver. Previously, I worked on the design and implementation of mine planning and scheduling systems, long term mine design with environmental focus, and mine planning corporate standards for Solid Energy, New Zealand. In addition, I have worked in various sales and support roles utilizing Vulcan Software and MineSuite Production Statistics where I gained considerable exposure to mining operations and projects around the world.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile Gold Mine property on May 9, 2017 for 2 days.
6. I am responsible for the preparation of Sections 15.1, 15.1.1, 15.1.2, 15.1.3, 16.1, 16.1.4, 16.1.6, [co-author 15.3, 16.3] and portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I havehad prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is conducting a high level due diligence of the property focusing on pit optimisation.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

"Signed" "Sealed"

Bret C. Swanson, B.Eng. (Mining), MAusIMM, MMSAQP[01418QP]

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY
234 St. Paul Street
Kamloops, BC Canada V2C 6G4
Telephone (250) 374-8600
Fax (250) 374-8606

CERTIFICATE OF QUALIFIED PERSON

I, Derek Kinakin, M.Sc., P.Geo., do hereby certify that:

1. I am an Engineering Geologist of BGC Engineering Inc., 234 St. Paul Street, Kamloops, B.C., Canada.
2. This certificate applies to the technical report titled "NI 43 101 Technical Report, Feasibility Study, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of January 1, 2017 (the "Technical Report").
3. I graduated with a degree in Physical Geography and Earth Science from Simon Fraser University in 2002. In addition, I have obtained a M.Sc. in Earth Science from Simon Fraser University in 2004. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC) and Alberta (APEGA). I have worked as an Engineering Geologist for a total of 14 years since my graduation from university. My relevant experience includes preliminary feasibility and feasibility studies of the open pit slopes of the Cortez Hills, KSM, Bisha Mine, and Donlin Creek projects. I provide ongoing geotechnical support for open pit mining at the Gibraltar, Bisha, and Cobre del Mayo mines.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile Gold Mine property on February 6, 2017 for three (3) days.
6. I am responsible for the preparation of Section 16.1.2 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was a geotechnical review of the initial pit slopes of the Millzone Pit.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

Derek Kinakin, M.Sc., P.Geo.

PROFESSIONAL
PROVINCE OF
D. KINAKIN
#32190
BRITISH COLUMBIA
GEOSCIENTIST
2017-08-09

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CERTIFICATE OF QUALIFIED PERSON

I, Grant Malensek, MEng, PEng/PGeo, do hereby certify that:

1. I am Principal Consultant (Mineral Economics) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Feasibility Study Haile Gold Mine Lancaster County, South Carolina" with an Effective Date of January 1, 2017 (the "Technical Report").
3. I graduated with a degree in B.S. Geological Sciences from University of British Columbia in 1987. In addition, I have obtained a M.E. in Geological Engineering (Colorado School of Mines, 1997) and a Graduate Business Certificate in Finance (University of Denver – Daniels College of Business, 2011). I hold a dual registration as a Professional Engineer/Professional Geoscientist of the Association of Professional Engineers & Geoscientists of British Columbia. I have worked as an Engineer for a total of over 20 years since my graduation from university. My relevant experience includes business experience in financial analysis, project management and business development.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Haile Gold Mine property.
6. I am responsible for the preparation of Sections 19, 19.1, 22, 22.1, 22.2, 22.2.1, 22.2.2, 22.2.3, 22.2.4, 22.2.5, 22.3, 22.4, 22.4.1, 22.4.2, 22.4.3, and portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report..
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

"Signed" "Sealed"

____

Grant Malensek, MEng, PEng/PGeo [APEGBC 23905]
Principal Consultant (Mineral Economics)

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CERTIFICATE OF QUALIFIED PERSON

I, David Bird, MSc., PG, RM-SME, do hereby certify that:

1. I am a Principal Consultant (Geochemistry) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report Feasibility Study Haile Gold Mine Lancaster County, South Carolina” with an Effective Date of January 1, 2017 (the “Technical Report”).
3. I graduated with Bachelor’s Degrees in Geology and Business Administration Management from Oregon State University in 1983. In addition, I obtained a Master’s Degree in Geochemistry/Hydrogeology from the University of Nevada-Reno in 1993. I am a Registered Member of the Society for Mining, Metallurgy, and Exploration (SME). I am a certified Professional Geologist in the State of Oregon (G1438). I have worked full time as a Geologist and Geochemist for a total of 32 years. My relevant experience includes design, execution, and interpretation of mine waste geochemical characterization programs in support of open pit and underground mine planning and environmental impact assessments, design and supervision of water quality sampling and monitoring programs, geochemical modeling, and management of the geochemistry portion of numerous PEA, PFS and FS-level mine projects in the US and abroad.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I have not visited the Haile Gold Mine property.
6. I am responsible for the preparation of Section 16.2.4, and portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

“Signed” “Sealed”

____

David Bird, MSc., PG, RM-SME[4186987RM]

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CERTIFICATE OF QUALIFIED PERSON

I, Bart A. Stryhas PhD, CPG, do hereby certify that:

1. I am a Principal Associate Resource Geologist of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report Feasibility Study Haile Gold Mine Lancaster County, South Carolina” with an Effective Date of January 1, 2017 (the “Technical Report”).
3. I graduated with a Doctorate degree in Structural Geology from Washington State University in 1988. In addition, I have obtained a Master of Science degree in Structural Geology from the University of Idaho in 1985 and a Bachelor of Arts degree in Geology from the University of Vermont in 1983. I am a current member of the American Institute of Professional Geologists. I have worked as a Geologist for a total of 27 years since my graduation from university. My relevant experience includes minerals exploration, mine geology, project development and resource estimation. I have conducted resource estimations since 1988 and have been involved in technical reports since 2004.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I visited the Haile Gold Mine property on October 10, 2016 for 3 days.
6. I am responsible for the preparation of Sections 14.2, 14.2.1, 14.2.2, 14.2.3, 14.2.4, 14.2.5, 14.2.6, 14.2.7, 14.2.8, 14.2.9, 14.2.10, 14.2.11, 14.2.12, 14.2.13, 14.2.14, co-author 14, 14.3 and portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

“Signed” “Sealed”

____

Bart A. Stryhas PhD, CPG #11034

Principal Associate Resource Geologist

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CERTIFICATE OF QUALIFIED PERSON

I, Brian S. Prosser, PE do hereby certify that:

1. I am Principal Consultant of SRK Consulting (U.S.), Inc., 1625 Shaw Avenue, #103, Clovis, CA, USA, 93611.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Feasibility Study Haile Gold Mine Lancaster County, South Carolina" with an Effective Date of January 1, 2017 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Virginia Polytechnic Institute and State University in 1994. In addition, I have obtained a licensure as a professional engineer in the states of Virginia and Nevada. I am a registered member of the Society for Mining, Metallurgy, and Exploration, Inc. I have worked as an engineer for a total of 22 years since my graduation from university. My relevant experience includes the design and optimization of subsurface ventilation systems for coal mines, metal mines, and repositories.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Haile Gold Mine property.
6. I am responsible for the preparation of Sections Section 16.2.10 and portions of Sections 1, 25 and 26 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of August, 2017.

"Signed" "Sealed"

Brian S. Prosser, PE

Appendix B: LoM Annual Cash Flow Forecast

Technical-Economic Model Summary

srk consulting

Company: **Oceanagold**
Business Unit: **Haile Optimisation Study**
Analysis Type: **FS**

YEAR			**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**2035**	**2036**
Time Before Mine Closure		**LoM**	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	-1	-2	-3	-4
Production Timeline		**Total**	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Discount Factors MOP @ 5%	US$ & Metric Units	**or Avg.**	0.9759	0.9294	0.8852	0.8430	0.8029	0.7646	0.7282	0.6936	0.6605	0.6291	0.5991	0.5706	0.5434	0.5175	0.4929	0.4694	0.4471	0.4258	0.4055	0.3862
Market Prices																						
Gold	US$/oz	**$1,300**	$1,300	$ 1,300	$ 1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300
Physicals																						
Total OP Ore Mined	kt	55,039	1,921	2,862	2,984	3,125	3,299	3,299	3,299	3,297	3,735	4,000	4,000	4,000	4,000	4,000	4,000	3,218	-	-	-	-
Total OP Waste Mined	kt	481,174	17,837	19,435	41,016	40,875	36,701	35,701	29,401	36,703	36,265	36,000	36,000	36,000	33,000	23,000	8,642	14,598	-	-	-	-
Total OP Material Mined	kt	536,213	19,758	22,297	44,000	44,000	40,000	39,000	32,700	40,000	40,000	40,000	40,000	40,000	37,000	27,000	12,642	17,816	-	-	-	-
Strip Ratio	w:o	8.7	9.3	6.8	13.7	13.1	11.1	10.8	8.9	11.1	9.7	9.0	9.0	9.0	8.2	5.8	2.2	4.5	-	-	-	-
Total UG Ore Mined	kt	3,116	-	-	-	45	701	701	701	703	265	-	-	-	-	-	-	-	-	-	-	-
Total Ore Tonnes, Mined	kt	58,155	1,921	2,862	2,984	3,170	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	3,218	-	-	-	-
Total Ore Tonnes, Processed	kt	58,155	1,805	2,500	3,000	3,260	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	4,000	3,590	-	-	-	-
Gold Grade, Processed	g/t	1.849	2.86	2.13	2.03	2.04	2.18	2.07	1.93	1.79	1.60	1.92	1.95	1.71	1.50	1.37	1.94	1.26	-	-	-	-
Contained Gold, Processed	koz	3,457	166	171	196	213	280	267	249	230	205	248	251	219	193	176	249	145	-	-	-	-
Average Recovery, Gold	%	82.5%	78.0%	83.7%	83.4%	83.4%	83.8%	83.5%	83.1%	82.6%	81.9%	83.1%	83.2%	82.3%	81.5%	80.8%	83.1%	80.2%	--	--	--	--
Recovered Gold, Dore	koz	2,854	129	143	163	178	235	223	207	190	168	206	209	181	157	142	207	116	-	-	-	-
Payable Gold, Dore	koz	2,854	129	143	163	178	235	223	207	190	168	206	209	181	157	142	207	116	-	-	-	-
Cash Flow																						
Gross Revenue	**$000s**	**3,709,892**	**168,032**	**185,916**	**212,038**	**231,558**	**305,090**	**289,511**	**268,865**	**247,052**	**218,552**	**267,355**	**271,302**	**234,909**	**204,107**	**184,964**	**269,296**	**151,344**	**-**	**-**	**-**	**-**
OP Mining Cost	$000s	(771,961)	(8,914)	(36,157)	(53,953)	(56,192)	(52,246)	(48,198)	(52,096)	(54,272)	(61,590)	(52,690)	(60,234)	(61,807)	(56,719)	(51,833)	(34,363)	(30,695)	-	-	-	-
UG Mining Cost	$000s	(120,133)	-	-	-	-	(27,856)	(25,743)	(25,936)	(28,903)	(11,696)	-	-	-	-	-	-	-	-	-	-	-
Process Cost	$000s	(596,491)	(6,045)	(28,995)	(33,120)	(35,269)	(41,370)	(41,370)	(41,370)	(41,370)	(41,370)	(41,370)	(41,370)	(41,370)	(41,370)	(41,370)	(41,370)	(37,987)	-	-	-	-
Site G&A Cost	$000s	(190,313)	(2,498)	(9,990)	(9,990)	(9,990)	(13,737)	(13,737)	(13,737)	(13,737)	(13,737)	(12,737)	(12,737)	(12,737)	(12,737)	(12,737)	(12,737)	(12,737)	-	-	-	-
Dore Refining/Freight Costs	$000s	(6,213)	(76)	(326)	(370)	(401)	(515)	(491)	(460)	(427)	(383)	(458)	(464)	(408)	(361)	(332)	(461)	(280)	-	-	-	-
By-Product Credit	$000s	68,280	3,273	3,374	3,862	4,216	5,528	5,264	4,914	4,543	4,055	4,888	4,955	4,335	3,807	3,477	4,921	2,867	-	-	-	-
Direct Cash Costs	**$000s**	**(1,616,832)**	**(14,260)**	**(72,095)**	**(93,573)**	**(97,636)**	**(130,196)**	**(124,275)**	**(128,685)**	**(134,167)**	**(124,722)**	**(102,367)**	**(109,850)**	**(111,988)**	**(107,381)**	**(102,796)**	**(84,009)**	**(78,833)**	**-**	**-**	**-**	**-**
Royalties	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Rehabilitation and Closure	$000s	(60,000)	-	-	(700)	(1,000)	(1,500)	(2,000)	(4,000)	(4,000)	(4,000)	(10,000)	(4,000)	(1,000)	(1,000)	(1,000)	(3,000)	(4,000)	(10,000)	(8,000)	(500)	(300)
Endowment Liability - State of SC	$000s	(9,686)	-	(1,514)	(1,514)	(1,514)	(514)	(514)	(514)	(514)	(514)	(514)	(514)	(514)	(514)	(514)	-	-	-	-	-	-
Interest Expense - Capital Leases	$000s	(2,500)	(309)	(1,054)	(753)	(366)	(17)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Principal Payment - Capital Leases	$000s	(29,697)	(1,739)	(7,140)	(7,442)	(12,772)	(274)	(330)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Regional Business Units	$000s	(25,954)	(966)	(2,601)	(2,390)	(1,818)	(1,818)	(1,818)	(1,818)	(1,818)	(1,818)	(1,818)	(1,818)	(1,818)	(1,818)	(1,818)	-	-	-	-	-	-
Indirect Cash Costs	**$000s**	**(127,837)**	**(3,014)**	**(12,310)**	**(12,798)**	**(17,470)**	**(4,124)**	**(4,663)**	**(6,332)**	**(6,332)**	**(6,332)**	**(12,332)**	**(6,332)**	**(3,332)**	**(3,332)**	**(3,332)**	**(3,000)**	**(4,000)**	**(10,000)**	**(8,000)**	**(500)**	**(300)**
Total Operating Expense	**$000s**	**(1,744,668)**	**(17,274)**	**(84,405)**	**(106,371)**	**(115,107)**	**(134,319)**	**(128,938)**	**(135,017)**	**(140,499)**	**(131,054)**	**(114,699)**	**(116,182)**	**(115,320)**	**(110,713)**	**(106,128)**	**(87,009)**	**(82,833)**	**(10,000)**	**(8,000)**	**(500)**	**(300)**
Operating Margin	**$000s**	**1,965,224**	**150,758**	**101,511**	**105,667**	**116,451**	**170,771**	**160,574**	**133,848**	**106,554**	**87,497**	**152,656**	**155,120**	**119,589**	**93,394**	**78,836**	**182,287**	**68,511**	**(10,000)**	**(8,000)**	**(500)**	**(300)**
Earnings Before Interest, Taxes & Depreciation	**$000s**	**1,965,224**	**150,758**	**101,511**	**105,667**	**116,451**	**170,771**	**160,574**	**133,848**	**106,554**	**87,497**	**152,656**	**155,120**	**119,589**	**93,394**	**78,836**	**182,287**	**68,511**	**(10,000)**	**(8,000)**	**(500)**	**(300)**
Depreciation Allowance	$000s	(967,965)	(45,852)	(49,776)	(60,838)	(67,590)	(75,660)	(81,426)	(81,411)	(83,213)	(82,806)	(85,791)	(43,643)	(38,957)	(37,904)	(37,647)	(51,732)	(43,718)	-	-	-	-
Depletion Allowance	$000s	(642,000)	(29,078)	(32,173)	(36,693)	(40,071)	(52,796)	(50,100)	(46,527)	(42,753)	(37,821)	(46,266)	(46,949)	(40,651)	(35,321)	(32,008)	(46,602)	(26,190)	-	-	-	-
Earnings Before Taxes	**$000s**	**355,259**	**75,827**	**19,562**	**8,136**	**8,790**	**42,315**	**29,047**	**5,909**	**(19,412)**	**(33,130)**	**20,599**	**64,528**	**39,981**	**20,169**	**9,181**	**83,953**	**(1,397)**	**(10,000)**	**(8,000)**	**(500)**	**(300)**
Income Tax @ Eff. Rate of 11.2%	$000s	(219,459)	(29,706)	(10,874)	(9,190)	(9,893)	(20,187)	(16,623)	(10,643)	(4,668)	(938)	(13,015)	(22,385)	(17,215)	(11,643)	(8,484)	(29,039)	(4,955)	-	-	-	-
Net Income	**$000s**	**135,800**	**46,122**	**8,688**	**(1,055)**	**(1,103)**	**22,128**	**12,424**	**(4,734)**	**(24,080)**	**(34,068)**	**7,584**	**42,143**	**22,767**	**8,526**	**696**	**54,914**	**(6,352)**	**(10,000)**	**(8,000)**	**(500)**	**(300)**
Non-Cash Add Back - Depreciation	$000s	967,965	45,852	49,776	60,838	67,590	75,660	81,426	81,411	83,213	82,806	85,791	43,643	38,957	37,904	37,647	51,732	43,718	-	-	-	-
Non-Cash Add Back - Depletion	$000s	642,000	29,078	32,173	36,693	40,071	52,796	50,100	46,527	42,753	37,821	46,266	46,949	40,651	35,321	32,008	46,602	26,190	-	-	-	-
Working Capital	$000s	-	(1,683)	(10)	(766)	(871)	(52)	213	282	305	383	(725)	(54)	505	415	262	(1,153)	1,617	1,333	-	-	-
Operating Cash Flow	**$000s**	**1,745,765**	**119,370**	**90,627**	**95,711**	**105,688**	**150,531**	**144,163**	**123,486**	**102,191**	**86,942**	**138,916**	**132,681**	**102,879**	**82,165**	**70,613**	**152,095**	**65,174**	**(8,667)**	**(8,000)**	**(500)**	**(300)**
Capitalized Preproduction Costs	$000s	(59,780)	(59,780)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
OP Mining	$000s	(59,928)	(20)	(3,172)	(48,517)	(8,219)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
UG Mining	$000s	(55,361)	-	(7)	(18,597)	(36,756)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Plant Expansion	$000s	(58,881)	-	-	(18,249)	(40,632)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure Expansion	$000s	(64,822)	(2,296)	(54,921)	(1,175)	(6,430)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	$000s	(15,725)	(202)	(581)	(7,998)	(6,943)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Development Capital	**$000s**	**(314,495)**	**(62,298)**	**(58,680)**	**(94,537)**	**(98,980)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
OP Mining	$000s	(73,068)	-	-	-	-	(1,202)	(1,440)	(4,050)	(6,441)	(18,167)	(4,460)	(10,149)	(3,412)	(9,502)	(11,629)	(2,616)	-	-	-	-	-
UG Mining	$000s	(25,696)	-	-	-	-	(9,075)	(6,807)	(9,339)	(474)	-	-	-	-	-	-	-	-	-	-	-	-
Plant (Sustaining)	$000s	(25,570)	(19,767)	(1,700)	(895)	(830)	(620)	(953)	(620)	(20)	(60)	(45)	-	(60)	-	-	-	-	-	-	-	-
Infrastructure Expansion	$000s	(119,551)	(10,190)	(9,950)	(1,680)	(2,630)	(2,200)	(16,420)	(1,650)	(24,912)	(1,550)	(33,175)	(500)	(500)	(8,356)	(4,500)	(1,338)	-	-	-	-	-
Contingency	$000s	(2,200)	-	-	-	-	(200)	(200)	(200)	(200)	(200)	(200)	(200)	(200)	(200)	(200)	(200)	-	-	-	-	-
Sustaining Capital	**$000s**	**(246,085)**	**(29,956)**	**(11,650)**	**(2,575)**	**(3,460)**	**(13,297)**	**(25,820)**	**(15,859)**	**(32,047)**	**(19,977)**	**(37,880)**	**(10,849)**	**(4,172)**	**(18,058)**	**(16,329)**	**(4,154)**	**-**	**-**	**-**	**-**	**-**
Total Capital	**$000s**	**(560,580)**	**(92,254)**	**(70,330)**	**(97,112)**	**(102,440)**	**(13,297)**	**(25,820)**	**(15,859)**	**(32,047)**	**(19,977)**	**(37,880)**	**(10,849)**	**(4,172)**	**(18,058)**	**(16,329)**	**(4,154)**	**-**	**-**	**-**	**-**	**-**

Technical-Economic Model Summary

srk consulting

Company: **Oceanagold**
Business Unit: **Haile Optimisation Study**
Analysis Type: **FS**

YEAR				2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Time Before Mine Closure			LoM	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	-1	-2	-3	-4
Production Timeline			Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Discount Factors	MOP @ 5%	US$ & Metric Units	or Avg.	0.9759	0.9294	0.8852	0.8430	0.8029	0.7646	0.7282	0.6936	0.6605	0.6291	0.5991	0.5706	0.5434	0.5175	0.4929	0.4694	0.4471	0.4258	0.4055	0.3862
Metrics																							
Economic Metrics																							
a) Pre-Tax																							
Free Cash Flow		$000s	**1,404,644**	56,821	31,170	7,790	13,140	157,421	134,967	118,271	74,812	67,903	114,051	144,217	115,921	75,750	62,769	176,979	70,128	(8,667)	(8,000)	(500)	(300)
Cumulative Free Cash Flow		$000s		56,821	87,992	95,781	108,922	266,343	401,310	519,580	594,393	662,296	776,347	920,564	1,036,485	1,112,235	1,175,003	1,351,982	1,422,111	1,413,444	1,405,444	1,404,944	1,404,644
NPV @	5.00%	$000s	**925,348**	55,452	28,971	6,895	11,078	126,390	103,201	86,128	51,886	44,852	71,746	86,403	66,143	41,164	32,485	87,232	32,920	(3,875)	(3,406)	(203)	(116)
Cumulative NPV		$000s		55,452	84,422	91,318	102,395	228,785	331,986	418,115	470,001	514,853	586,600	673,002	739,146	780,310	812,795	900,027	932,947	929,072	925,666	925,463	925,348
IRR		%	**NA**																				
PI @	5.00%	NPV / (PW of TC)	**2.02**	90,031	65,367	85,960	86,359	10,676	19,743	11,549	22,226	13,195	23,830	6,500	2,380	9,813	8,451	2,048	-	-	-	-	-
b) After-Tax																							
Free Cash Flow		$000s	**1,185,185**	27,116	20,297	(1,401)	3,247	137,234	118,343	107,627	70,144	66,965	101,036	121,832	98,707	64,106	54,284	147,940	65,174	(8,667)	(8,000)	(500)	(300)
Cumulative Free Cash Flow		$000s		27,116	47,412	46,012	49,259	186,493	304,836	412,463	482,608	549,573	650,608	772,440	871,147	935,254	989,538	1,137,478	1,202,652	1,193,985	1,185,985	1,185,485	1,185,185
NPV @	5.00%	$000s	**770,471**	26,462	18,864	(1,240)	2,737	110,182	90,490	78,378	48,649	44,232	63,559	72,992	56,321	34,837	28,094	72,919	30,594	(3,875)	(3,406)	(203)	(116)
Cumulative NPV		$000s		26,462	45,326	44,087	46,824	157,006	247,496	325,874	374,523	418,755	482,314	555,306	611,627	646,463	674,558	747,477	778,071	774,196	770,790	770,587	770,471
IRR		%	**NA**																				
PI @	5.00%	NPV / (PW of TC)	**1.68**	90,031	65,367	85,960	86,359	10,676	19,743	11,549	22,226	13,195	23,830	6,500	2,380	9,813	8,451	2,048	-	-	-	-	-
Operating Metrics (RoM)																							
Mine Life		Years	16																				
Maximum Mining Rate (Ore + Waste)		MTPA	44,000																				
Maximum Ore Processing Rate		MTPA	4,000																				
OP Mining Cost		$ / t rock	$1.48	1.74	1.62	1.23	1.28	1.31	1.24	1.59	1.36	1.54	1.32	1.51	1.55	1.53	1.92	2.72	1.72	-	-	-	-
UG Mining Cost		$ / t UG ore mined	$39.11	-	-	-	-	39.73	36.72	36.99	41.10	44.14	-	-	-	-	-	-	-	-	-	-	-
Total Mining Cost		$ / t ore milled	$15.71	15.80	12.63	18.08	17.73	20.03	18.49	19.51	20.79	18.32	13.17	15.06	15.45	14.18	12.96	8.59	9.54	-	-	-	-
Processing Cost		$ / t ore milled	$10.50	12.62	11.60	11.04	10.82	10.34	10.34	10.34	10.34	10.34	10.34	10.34	10.34	10.34	10.34	10.34	10.58	-	-	-	-
G&A Cost	13%	$ / t ore milled	$3.35	5.22	4.00	3.33	3.06	3.43	3.43	3.43	3.43	3.43	3.18	3.18	3.18	3.18	3.18	3.18	3.55	-	-	-	-
Dore Refining/Freight Costs		$ / t ore milled	$0.11	0.16	0.13	0.12	0.12	0.13	0.12	0.11	0.11	0.10	0.11	0.12	0.10	0.09	0.08	0.12	0.08	-	-	-	-
Indirect Costs		$ / t ore milled	$2.25	6.29	4.92	4.27	5.36	1.03	1.17	1.58	1.58	1.58	3.08	1.58	0.83	0.83	0.83	0.75	1.11	-	-	-	-
Total Operating Costs		$ / t ore milled	$31.91	40.09	33.28	36.84	37.09	34.96	33.55	34.98	36.26	33.78	29.90	30.28	29.91	28.63	27.40	22.98	24.86	-	-	-	-
Sales Metrics																							
LoM Dore Sales		koz Au	2,854	129	143	163	178	235	223	207	190	168	206	209	181	157	142	207	116	-	-	-	-
Less Preproduction Sales		koz Au	(97)	(97)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
RoM Dore Sales		koz Au	2,756	32	143	163	178	235	223	207	190	168	206	209	181	157	142	207	116	-	-	-	-
LoM All-In Sustaining Costs (AISC)		$000s	2,040,496	107,010	96,055	108,946	122,950	147,617	140,338	150,876	172,546	151,031	152,579	127,031	119,492	128,771	122,457	91,164	82,833	10,000	8,000	500	300
Less Preproduction Costs		$000s	(89,802)	(85,419)	-	-	(4,383)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
RoM All-In Sustaining Costs (AISC)		$000s	1,931,894	21,591	96,055	108,946	118,567	147,617	140,338	150,876	172,546	151,031	152,579	127,031	119,492	128,771	122,457	91,164	82,833	-	-	-	-
LoM AISC Metric		$ / oz Au	$715	$828	$672	$668	$690	$629	$630	$730	$908	$898	$742	$609	$661	$820	$861	$440	$712	$0	$0	$0	$0
Less Preproduction AISC Metric		$ / oz Au	$923	$878	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
RoM AISC Metric		$ / oz Au	$701	$675	$672	$668	$666	$629	$630	$730	$908	$898	$742	$609	$661	$820	$861	$440	$712	$0	$0	$0	$0
Phase 1 OP Only (Oct 2017-2020)		$ / oz Au	$669																				
Phase 2 OP+UG (2021-2025)		$ / oz Au	$746																				
Phase 3 OP Only (2026-2032 EoM)		$ / oz Au	$677																				

Appendix C: Taxation Model

YEAR				2017	2017	2017	2017	2017	2017	2017	2017	2017	2017	2017	2017
Quarter				1Q2017	1Q2017	1Q2017	2Q2017	2Q2017	2Q2017	3Q2017	3Q2017	3Q2017	4Q2017	4Q2017	4Q2017
Month				Jan-17	Feb-17	Mar-17	Apr-17	May-17	Jun-17	Jul-17	Aug-17	Sep-17	Oct-17	Nov-17	Dec-17
Days Per Period				31	28	31	30	31	30	31	31	30	31	30	31
Year Counter			LoM Total	1	1	1	1	1	1	1	1	1	1	1	1
Project Timeline	EOP @ 5%	US$ & Metric Units	or Avg.	1	2	3	4	5	6	7	8	9	10	11	12
INCOME TAX															
Standard Federal CIT															
Gross Gold Revenue		$000s	3,709,892	5,183	13,225	14,213	15,556	16,719	16,173	15,908	15,259	14,236	14,044	13,562	13,954
Silver By-Product Credit		$000s	68,280	103	258	277	302	324	313	309	297	278	275	265	273
Selling/Refining Costs		$000s	(6,213)	-	-	-	-	-	-	-	-	-	(26)	(25)	(26)
Operating Costs		$000s	(1,678,898)	-	-	-	-	-	-	-	-	-	(5,834)	(5,789)	(5,834)
Indirect Costs		$000s	(127,837)	-	-	-	-	-	-	-	-	-	(2,189)	(350)	(476)
Adjusted EBITDA		**$000s**	**1,965,224**	**5,286**	**13,483**	**14,490**	**15,858**	**17,042**	**16,486**	**16,217**	**15,556**	**14,514**	**6,271**	**7,663**	**7,892**
Prior Initial Construction Depreciation		$000s	(417,000)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)
Calculated Project Depreciation		$000s	(550,965)	(128)	(327)	(351)	(384)	(413)	(400)	(393)	(377)	(352)	(347)	(335)	(345)
Depletion		$000s	(642,000)	(897)	(2,289)	(2,460)	(2,692)	(2,893)	(2,799)	(2,753)	(2,641)	(2,464)	(2,430)	(2,347)	(2,415)
DPA Deduction		$000s	(33,913)	(395)	(464)	(396)	(364)	(419)	(431)	(422)	(455)	(443)	-	(138)	(148)
Earnings Before Taxes		**$000s**	**321,346**	**391**	**6,929**	**7,808**	**8,943**	**9,842**	**9,381**	**9,174**	**8,609**	**7,780**	**18**	**1,369**	**1,509**
Loss Carry Forward		$000s	(59,627)	-	-	-	-	-	-	-	-	-	-	-	-
Net Taxable Income / (NOL)		**$000s**	**261,719**	**391**	**6,929**	**7,808**	**8,943**	**9,842**	**9,381**	**9,174**	**8,609**	**7,780**	**18**	**1,369**	**1,509**
Standard Federal Income Tax	35%	**$000s**	**119,547**	**137**	**2,425**	**2,733**	**3,130**	**3,445**	**3,283**	**3,211**	**3,013**	**2,723**	**6**	**479**	**528**
Federal Alt. Minimum Income Tax	YES														
Taxable Income Before NOL		$000s	321,346	391	6,929	7,808	8,943	9,842	9,381	9,174	8,609	7,780	18	1,369	1,509
Depletion Add back		$000s	642,000	897	2,289	2,460	2,692	2,893	2,799	2,753	2,641	2,464	2,430	2,347	2,415
AMTI Before NOL		$000s	963,346	1,288	9,218	10,267	11,635	12,735	12,180	11,927	11,250	10,244	2,449	3,716	3,923
AMTI After NOL	0	$000s	982,146	1,288	9,218	10,267	11,635	12,735	12,180	11,927	11,250	10,244	2,449	3,716	3,923
Alternative Minimum Income Tax	20%	**$000s**	**196,429**	**258**	**1,844**	**2,053**	**2,327**	**2,547**	**2,436**	**2,385**	**2,250**	**2,049**	**490**	**743**	**785**
Federal CIT Reconciliation															
Regular Federal Income Tax After NOL		$000s	119,547	137	2,425	2,733	3,130	3,445	3,283	3,211	3,013	2,723	6	479	528
AMT After NOL	0	$000s	82,834	121	(121)	-	-	-	-	-	-	-	483	264	257
Adjusted Federal Income Tax		**$000s**	**202,381**	**258**	**2,304**	**2,733**	**3,130**	**3,445**	**3,283**	**3,211**	**3,013**	**2,723**	**490**	**743**	**785**
State Income Tax															
South Carolina Income Tax	5%	$000s	**17,078**	**20**	**346**	**390**	**447**	**492**	**469**	**459**	**430**	**389**	**1**	**68**	**75**
Total Corporate Income Tax		**$000s**	**219,459**	**277**	**2,651**	**3,123**	**3,577**	**3,937**	**3,752**	**3,670**	**3,444**	**3,112**	**491**	**812**	**860**
Loss Carry Forward															
Opening Balance	0	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-
Additions		$000s	(59,627)	-	-	-	-	-	-	-	-	-	-	-	-
Losses Used		$000s	59,627	-	-	-	-	-	-	-	-	-	-	-	-
Closing Balance		$000s	(20,217)	-	-	-	-	-	-	-	-	-	-	-	-
Depletion															
1) Percentage Depletion															
Gross Income From Mining Method															
Gross Revenue		$000s	3,709,892	5,183	13,225	14,213	15,556	16,719	16,173	15,908	15,259	14,236	14,044	13,562	13,954
Less Royalty		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-
Adj. Gross Income		**$000s**	**3,709,892**	5,183	13,225	14,213	15,556	16,719	16,173	15,908	15,259	14,236	14,044	13,562	13,954
GIFM Depletion	15%	**$000s**	**399,791**	**777**	**1,984**	**2,132**	**2,333**	**2,508**	**2,426**	**2,386**	**2,289**	**2,135**	**2,107**	**2,034**	**2,093**
Net Income From Mining Method															
Gross Income		$000s	3,709,892	5,183	13,225	14,213	15,556	16,719	16,173	15,908	15,259	14,236	14,044	13,562	13,954
Less Operating Costs		$000s	(1,812,948)	-	-	-	-	-	-	-	-	-	(8,048)	(6,164)	(6,335)
Less Depreciation		$000s	(967,965)	(3,603)	(3,802)	(3,826)	(3,859)	(3,888)	(3,875)	(3,868)	(3,852)	(3,827)	(3,822)	(3,810)	(3,820)
Taxable Income Before Depletion		**$000s**	**928,979**	1,580	9,423	10,386	11,697	12,831	12,298	12,040	11,407	10,409	2,174	3,588	3,799
NIFM Limit	50%	**$000s**	**317,263**	**790**	**4,712**	**5,193**	**5,848**	**6,415**	**6,149**	**6,020**	**5,704**	**5,205**	**1,087**	**1,794**	**1,899**
Smaller Allowed Percent Depletion		**$000s**	**405,354**	**777**	**1,984**	**2,132**	**2,333**	**2,508**	**2,426**	**2,386**	**2,289**	**2,135**	**1,087**	**1,794**	**1,899**
2) Cost Depletion															
Acquisition Cost Basis	642,000	$000s	-	642,000	641,103	638,814	636,355	633,663	630,770	627,971	625,218	622,577	620,114	617,684	615,337
LoM Recovered Ounces	2,854	koz	2,854	4	10	11	12	13	12	12	12	11	11	10	11
Cost Depletion Allowed		**$000s**	**642,000**	**897**	**2,289**	**2,460**	**2,692**	**2,893**	**2,799**	**2,753**	**2,641**	**2,464**	**2,430**	**2,347**	**2,415**
Larger of Allowed Depletion		**$000s**	**642,000**	**897**	**2,289**	**2,460**	**2,692**	**2,893**	**2,799**	**2,753**	**2,641**	**2,464**	**2,430**	**2,347**	**2,415**
Domestic Productions Activities Deduction															
EBITDA															
Net Free Cash Flow before Tax		$000s	1,533,576	(1,958)	19,968	22,588	19,091	23,093	22,135	21,377	22,228	19,565	3,012	7,366	7,288
Capital Cost Addition		$000s	560,580	12,546	7,028	6,390	12,631	10,994	10,836	11,055	8,882	9,460	1,515	321	594
Depreciation Deduction		$000s	(967,965)	(3,603)	(3,802)	(3,826)	(3,859)	(3,888)	(3,875)	(3,868)	(3,852)	(3,827)	(3,822)	(3,810)	(3,820)
Depletion Deduction		**$000s**	(642,000)	(897)	(2,289)	(2,460)	(2,692)	(2,893)	(2,799)	(2,753)	(2,641)	(2,464)	(2,430)	(2,347)	(2,415)
Net Taxable Income / (NOL)		**$000s**	**484,191**	**6,088**	**20,905**	**22,693**	**25,171**	**27,305**	**26,298**	**25,811**	**24,618**	**22,735**	**(1,725)**	**1,530**	**1,648**
Net Taxable Income Amount	9%	**$000s**	50,644	548	1,881	2,042	2,265	2,457	2,367	2,323	2,216	2,046	-	138	148
Mining Salaries and Wages Amount	50%	**$000s**	153,200	395	464	396	364	419	431	422	455	443	443	443	443
DPA Deduction	YES	**$000s**	**33,913**	**395**	**464**	**396**	**364**	**419**	**431**	**422**	**455**	**443**	**-**	**138**	**148**
Depreciation															
1) Straight Line															
a) Undepreciated Initial Construction															
Undepreciated Balance as of Jan 2017	417,000	$000s													
Depreciation Term	10	Years													
Initial Construction Depreciation		**$000s**	**417,000**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**
b) Mobile Equipment - Project															
Total OP+UG Mobile Equipment Capex		$000s	150,770	-	-	-	-	-	-	-	-	-	-	-	-
Calendar Year		-		2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
Total Capex by Year		$000s	150,770	-	-	62,063	18,118	4,739	1,952	3,459	5,053	17,965	4,200	9,957	-
Depreciation Term	7	Years													
Total Annual Depreciation By Year	0	$000s	150,770	-	-	8,866	11,454	12,131	12,410	12,904	13,626	16,193	7,927	6,761	6,084
Total MC Depreciation by Period		$000s	150,770	-	-	-	-	-	-	-	-	-	-	-	-

YEAR	2018	2018	2018	2018	2018	2018	2018	2018	2018	2018	2018	2018	2019	2019	2019	2019	2020	2020	2020	2020
Quarter	1Q2018	1Q2018	1Q2018	2Q2018	2Q2018	2Q2018	3Q2018	3Q2018	3Q2018	4Q2018	4Q2018	4Q2018	1Q2019	2Q2019	3Q2019	4Q2019	1Q2020	2Q2020	3Q2020	4Q2020
Month	Jan-18	Feb-18	Mar-18	Apr-18	May-18	Jun-18	Jul-18	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18	Mar-19	Jun-19	Sep-19	Dec-19	Mar-20	Jun-20	Sep-20	Dec-20
Days Per Period	31	28	31	30	31	30	31	31	30	31	30	31	90	91	92	92	91	91	92	92
Year Counter	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	4	4	4	4
Project Timeline	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32
INCOME TAX																				
Standard Federal CIT																				
Gross Gold Revenue	19,046	16,248	17,191	15,737	15,835	14,911	15,157	15,404	14,493	14,373	13,503	14,017	50,957	52,260	52,992	55,830	53,529	52,906	53,249	71,874
Silver By-Product Credit	342	292	310	285	287	271	276	280	264	263	247	257	930	953	966	1,014	974	964	970	1,308
Selling/Refining Costs	(33)	(28)	(30)	(28)	(28)	(26)	(27)	(27)	(26)	(25)	(24)	(25)	(89)	(91)	(93)	(97)	(93)	(92)	(93)	(122)
Operating Costs	(6,058)	(5,884)	(6,058)	(6,000)	(6,363)	(6,305)	(6,363)	(6,524)	(6,466)	(6,454)	(6,305)	(6,363)	(24,180)	(24,249)	(24,318)	(24,318)	(24,808)	(24,808)	(24,878)	(26,957)
Indirect Costs	(3,170)	(317)	(221)	(2,183)	(824)	(213)	(2,173)	(319)	(214)	(2,156)	(317)	(204)	(3,865)	(3,337)	(2,804)	(2,792)	(3,748)	(3,285)	(9,648)	(789)
Adjusted EBITDA	**10,127**	**10,310**	**11,193**	**7,811**	**8,908**	**8,638**	**6,870**	**8,815**	**8,052**	**6,000**	**7,104**	**7,682**	**23,752**	**25,535**	**26,742**	**29,638**	**25,853**	**25,684**	**19,601**	**45,314**
Prior Initial Construction Depreciation	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(3,475)	(10,425)	(10,425)	(10,425)	(10,425)	(10,425)	(10,425)	(10,425)	(10,425)
Calculated Project Depreciation	(827)	(706)	(747)	(684)	(688)	(648)	(658)	(669)	(630)	(624)	(587)	(609)	(4,685)	(4,748)	(4,784)	(4,921)	(6,201)	(6,162)	(6,183)	(7,344)
Depletion	(3,296)	(2,812)	(2,975)	(2,723)	(2,740)	(2,580)	(2,623)	(2,666)	(2,508)	(2,487)	(2,337)	(2,426)	(8,818)	(9,044)	(9,170)	(9,662)	(9,263)	(9,156)	(9,215)	(12,438)
DPA Deduction	(154)	(319)	(366)	(99)	(185)	(182)	(12)	(177)	(137)	-	(71)	(103)	-	(114)	(211)	(403)	(6)	-	-	(1,269)
Earnings Before Taxes	**2,375**	**2,998**	**3,630**	**830**	**1,820**	**1,753**	**102**	**1,828**	**1,302**	**(586)**	**635**	**1,070**	**(176)**	**1,204**	**2,152**	**4,227**	**(42)**	**(59)**	**(6,223)**	**13,838**
Loss Carry Forward	-	-	-	-	-	-	-	-	-	-	(586)	-	-	(176)	-	-	-	-	-	(6,323)
Net Taxable Income / (NOL)	**2,375**	**2,998**	**3,630**	**830**	**1,820**	**1,753**	**102**	**1,828**	**1,302**	**(586)**	**48**	**1,070**	**(176)**	**1,028**	**2,152**	**4,227**	**(42)**	**(59)**	**(6,223)**	**7,515**
Standard Federal Income Tax	**831**	**1,049**	**1,271**	**291**	**637**	**613**	**36**	**640**	**456**	**-**	**17**	**374**	**-**	**360**	**753**	**1,479**	**-**	**-**	**-**	**2,630**
Federal Alt. Minimum Income Tax																				
Taxable Income Before NOL	2,375	2,998	3,630	830	1,820	1,753	102	1,828	1,302	(586)	635	1,070	(176)	1,204	2,152	4,227	(42)	(59)	(6,223)	13,838
Depletion Add back	3,296	2,812	2,975	2,723	2,740	2,580	2,623	2,666	2,508	2,487	2,337	2,426	8,818	9,044	9,170	9,662	9,263	9,156	9,215	12,438
AMTI Before NOL	5,671	5,810	6,605	3,554	4,560	4,333	2,725	4,494	3,810	1,901	2,971	3,495	8,642	10,247	11,322	13,889	9,222	9,097	2,992	26,276
AMTI After NOL	5,671	5,810	6,605	3,554	4,560	4,333	2,725	4,494	3,810	1,901	2,971	3,495	8,642	10,247	11,322	13,889	9,222	9,097	2,992	26,276
Alternative Minimum Income Tax	**1,134**	**1,162**	**1,321**	**711**	**912**	**867**	**545**	**899**	**762**	**380**	**594**	**699**	**1,728**	**2,049**	**2,264**	**2,778**	**1,844**	**1,819**	**598**	**5,255**
Federal CIT Reconciliation																				
Regular Federal Income Tax After NOL	831	1,049	1,271	291	637	613	36	640	456	-	17	374	-	360	753	1,479	-	-	-	2,630
AMT After NOL	303	113	50	420	275	253	509	259	306	380	577	325	1,728	1,690	1,511	1,298	1,844	1,819	598	2,625
Adjusted Federal Income Tax	**1,134**	**1,162**	**1,321**	**711**	**912**	**867**	**545**	**899**	**762**	**380**	**594**	**699**	**1,728**	**2,049**	**2,264**	**2,778**	**1,844**	**1,819**	**598**	**5,255**
State Income Tax																				
South Carolina Income Tax	**119**	**150**	**182**	**42**	**91**	**88**	**5**	**91**	**65**	**-**	**2**	**53**	**-**	**51**	**108**	**211**	**-**	**-**	**-**	**376**
Total Corporate Income Tax	**1,253**	**1,312**	**1,503**	**752**	**1,003**	**954**	**550**	**990**	**827**	**380**	**597**	**753**	**1,728**	**2,101**	**2,372**	**2,989**	**1,844**	**1,819**	**598**	**5,631**
Loss Carry Forward																				
Opening Balance	-	-	-	-	-	-	-	-	-	-	(586)	-	-	(176)	-	-	-	(42)	(100)	(6,323)
Additions	-	-	-	-	-	-	-	-	-	(586)	-	-	(176)	-	-	-	(42)	(59)	(6,223)	-
Losses Used	-	-	-	-	-	-	-	-	-	-	586	-	-	176	-	-	-	-	-	6,323
Closing Balance	-	-	-	-	-	-	-	-	-	(586)	-	-	(176)	-	-	-	(42)	(100)	(6,323)	-
Depletion																				
1) Percentage Depletion																				
Gross Income From Mining Method																				
Gross Revenue	19,046	16,248	17,191	15,737	15,835	14,911	15,157	15,404	14,493	14,373	13,503	14,017	50,957	52,260	52,992	55,830	53,529	52,906	53,249	71,874
Less Royalty	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Adj. Gross Income	19,046	16,248	17,191	15,737	15,835	14,911	15,157	15,404	14,493	14,373	13,503	14,017	50,957	52,260	52,992	55,830	53,529	52,906	53,249	71,874
GIFM Depletion	**2,857**	**2,437**	**2,579**	**2,361**	**2,375**	**2,237**	**2,274**	**2,311**	**2,174**	**2,156**	**2,025**	**2,103**	**7,643**	**7,839**	**7,949**	**8,375**	**8,029**	**7,936**	**7,987**	**10,781**
Net Income From Mining Method																				
Gross Income	19,046	16,248	17,191	15,737	15,835	14,911	15,157	15,404	14,493	14,373	13,503	14,017	50,957	52,260	52,992	55,830	53,529	52,906	53,249	71,874
Less Operating Costs	(9,261)	(6,230)	(6,309)	(8,211)	(7,214)	(6,545)	(8,562)	(6,869)	(6,705)	(8,635)	(6,646)	(6,592)	(28,134)	(27,677)	(27,215)	(27,207)	(28,650)	(28,186)	(34,619)	(27,868)
Less Depreciation	(4,302)	(4,181)	(4,222)	(4,159)	(4,163)	(4,123)	(4,133)	(4,144)	(4,105)	(4,099)	(4,062)	(4,084)	(15,110)	(15,173)	(15,209)	(15,346)	(16,626)	(16,587)	(16,608)	(17,769)
Taxable Income Before Depletion	5,483	5,837	6,661	3,368	4,458	4,244	2,461	4,391	3,683	1,638	2,795	3,342	7,713	9,409	10,568	13,278	8,253	8,133	2,022	26,236
NIFM Limit	**2,742**	**2,919**	**3,330**	**1,684**	**2,229**	**2,122**	**1,230**	**2,195**	**1,842**	**819**	**1,398**	**1,671**	**3,856**	**4,705**	**5,284**	**6,639**	**4,127**	**4,067**	**1,011**	**13,118**
Smaller Allowed Percent Depletion	**2,742**	**2,437**	**2,579**	**1,684**	**2,229**	**2,122**	**1,230**	**2,195**	**1,842**	**819**	**1,398**	**1,671**	**3,856**	**4,705**	**5,284**	**6,639**	**4,127**	**4,067**	**1,011**	**10,781**
2) Cost Depletion																				
Acquisition Cost Basis	612,922	609,626	606,814	603,839	601,116	598,376	595,795	593,172	590,507	587,999	585,511	583,175	580,749	571,931	562,887	553,717	544,056	534,792	525,637	516,422
LoM Recovered Ounces	15	12	13	12	12	11	12	12	11	11	10	11	39	40	41	43	41	41	41	55
Cost Depletion Allowed	**3,296**	**2,812**	**2,975**	**2,723**	**2,740**	**2,580**	**2,623**	**2,666**	**2,508**	**2,487**	**2,337**	**2,426**	**8,818**	**9,044**	**9,170**	**9,662**	**9,263**	**9,156**	**9,215**	**12,438**
Larger of Allowed Depletion	**3,296**	**2,812**	**2,975**	**2,723**	**2,740**	**2,580**	**2,623**	**2,666**	**2,508**	**2,487**	**2,337**	**2,426**	**8,818**	**9,044**	**9,170**	**9,662**	**9,263**	**9,156**	**9,215**	**12,438**
Domestic Productions Activities Deduction																				
EBITDA																				
Net Free Cash Flow before Tax	4,138	8,582	(7,184)	7,303	8,349	(5,884)	6,238	8,613	(5,884)	6,015	7,027	(6,141)	(34,080)	20,551	7,322	13,998	(9,566)	(349)	532	22,524
Capital Cost Addition	5,169	1,959	18,445	677	609	14,609	654	162	14,024	64	164	13,794	57,282	4,938	19,406	15,486	35,518	26,067	19,076	21,779
Depreciation Deduction	(4,302)	(4,181)	(4,222)	(4,159)	(4,163)	(4,123)	(4,133)	(4,144)	(4,105)	(4,099)	(4,062)	(4,084)	(15,110)	(15,173)	(15,209)	(15,346)	(16,626)	(16,587)	(16,608)	(17,769)
Depletion Deduction	(3,296)	(2,812)	(2,975)	(2,723)	(2,740)	(2,580)	(2,623)	(2,666)	(2,508)	(2,487)	(2,337)	(2,426)	(8,818)	(9,044)	(9,170)	(9,662)	(9,263)	(9,156)	(9,215)	(12,438)
Net Taxable Income / (NOL)	**1,708**	**3,549**	**4,064**	**1,098**	**2,055**	**2,022**	**136**	**1,965**	**1,527**	**(508)**	**792**	**1,143**	**(726)**	**1,271**	**2,349**	**4,476**	**63**	**(25)**	**(6,216)**	**14,096**
Net Taxable Income Amount	154	319	366	99	185	182	12	177	137	-	71	103	-	114	211	403	6	-	-	1,269
Mining Salaries and Wages Amount	412	412	412	412	458	458	458	482	482	471	458	458	2,010	2,010	2,010	2,010	2,092	2,092	2,092	3,378
DPA Deduction	**154**	**319**	**366**	**99**	**185**	**182**	**12**	**177**	**137**	**-**	**71**	**103**	**-**	**114**	**211**	**403**	**6**	**-**	**-**	**1,269**
Depreciation																				
1) Straight Line																				
a) Undepreciated Initial Construction																				
Undepreciated Balance as of Jan 2017																				
Depreciation Term																				
Initial Construction Depreciation	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**3,475**	**10,425**	**10,425**	**10,425**	**10,425**	**10,425**	**10,425**	**10,425**	**10,425**
b) Mobile Equipment - Project																				
Total OP+UG Mobile Equipment Capex	-	-	-	-	-	-	-	-	-	-	-	-	53,210	-	8,853	-	10,398	-	1,317	6,403
Calendar Year	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048
Total Capex by Year	9,300	11,465	2,500	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Depreciation Term																				
Total Annual Depreciation By Year	7,133	8,277	7,912	5,346	4,746	3,324	3,324	1,995	357	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)
Total MC Depreciation by Period	-	-	-	-	-	-	-	-	-	-	-	-	2,217	2,217	2,217	2,217	2,864	2,864	2,864	2,864

YEAR				2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Quarter				4Q2021	4Q2022	4Q2023	4Q2024	4Q2025	4Q2026	4Q2027	4Q2028	4Q2029	4Q2030	4Q2031	4Q2032	4Q2033	4Q2034	4Q2035	4Q2036	4Q2037
Month				Dec-21	Dec-22	Dec-23	Dec-24	Dec-25	Dec-26	Dec-27	Dec-28	Dec-29	Dec-30	Dec-31	Dec-32	Dec-33	Dec-34	Dec-35	Dec-36	Dec-37
Days Per Period				365	365	365	366	365	365	365	366	365	365	365	366	365	365	365	366	365
Year Counter			LoM Total	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
Project Timeline	EOP @ 5%	US$ & Metric Units	or Avg.	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49
INCOME TAX																				
Standard Federal CIT																				
Gross Gold Revenue		$000s	3,709,892	305,090	289,511	268,865	247,052	218,552	267,355	271,302	234,909	204,107	184,964	269,296	151,344	-	-	-	-	-
Silver By-Product Credit		$000s	68,280	5,528	5,264	4,914	4,543	4,055	4,888	4,955	4,335	3,807	3,477	4,921	2,867	-	-	-	-	-
Selling/Refining Costs		$000s	(6,213)	(515)	(491)	(460)	(427)	(383)	(458)	(464)	(408)	(361)	(332)	(461)	(280)	-	-	-	-	-
Operating Costs		$000s	(1,678,898)	(135,209)	(129,048)	(133,139)	(138,282)	(128,394)	(106,797)	(114,342)	(115,914)	(110,826)	(105,941)	(88,470)	(81,420)	-	-	-	-	-
Indirect Costs		$000s	(127,837)	(4,124)	(4,663)	(6,332)	(6,332)	(6,332)	(12,332)	(6,332)	(3,332)	(3,332)	(3,332)	(3,000)	(4,000)	(10,000)	(8,000)	(500)	(300)	-
Adjusted EBITDA		**$000s**	**1,965,224**	**170,771**	**160,574**	**133,848**	**106,554**	**87,497**	**152,656**	**155,120**	**119,589**	**93,394**	**78,836**	**182,287**	**68,511**	**(10,000)**	**(8,000)**	**(500)**	**(300)**	**-**
Prior Initial Construction Depreciation		$000s	(417,000)	(41,700)	(41,700)	(41,700)	(41,700)	(41,700)	(41,700)	-	-	-	-	-	-	-	-	-	-	-
Calculated Project Depreciation		$000s	(550,965)	(33,960)	(39,726)	(39,711)	(41,513)	(41,106)	(44,091)	(43,643)	(38,957)	(37,904)	(37,647)	(51,732)	(43,718)	-	-	-	-	-
Depletion		$000s	(642,000)	(52,796)	(50,100)	(46,527)	(42,753)	(37,821)	(46,266)	(46,949)	(40,651)	(35,321)	(32,008)	(46,602)	(26,190)	-	-	-	-	-
DPA Deduction		$000s	(33,913)	(3,804)	(2,633)	(557)	-	-	(1,789)	(5,803)	(3,644)	(1,853)	(850)	(5,079)	(20)	-	-	-	-	-
Earnings Before Taxes		**$000s**	**321,346**	**38,512**	**26,414**	**5,352**	**(19,412)**	**(33,130)**	**18,811**	**58,725**	**36,337**	**18,317**	**8,331**	**78,873**	**(1,417)**	**(10,000)**	**(8,000)**	**(500)**	**(300)**	**-**
Loss Carry Forward		$000s	(59,627)	-	-	-	-	-	(18,811)	(33,731)	-	-	-	-	-	-	-	-	-	-
Net Taxable Income / (NOL)		**$000s**	**261,719**	**38,512**	**26,414**	**5,352**	**(19,412)**	**(33,130)**	**-**	**24,995**	**36,337**	**18,317**	**8,331**	**78,873**	**(1,417)**	**(10,000)**	**(8,000)**	**(500)**	**(300)**	**-**
Standard Federal Income Tax	35%	**$000s**	**119,547**	**13,479**	**9,245**	**1,873**	**-**	**-**	**-**	**8,748**	**12,718**	**6,411**	**2,916**	**27,606**	**-**	**-**	**-**	**-**	**-**	**-**
Federal Alt. Minimum Income Tax	**YES**																			
Taxable Income Before NOL		$000s	321,346	38,512	26,414	5,352	(19,412)	(33,130)	18,811	58,725	36,337	18,317	8,331	78,873	(1,417)	(10,000)	(8,000)	(500)	(300)	-
Depletion Add back		$000s	642,000	52,796	50,100	46,527	42,753	37,821	46,266	46,949	40,651	35,321	32,008	46,602	26,190	-	-	-	-	-
AMTI Before NOL		$000s	963,346	91,308	76,514	51,879	23,341	4,691	65,077	105,674	76,989	53,638	40,339	125,475	24,774	(10,000)	(8,000)	(500)	(300)	-
AMTI After NOL	0	$000s	982,146	91,308	76,514	51,879	23,341	4,691	65,077	105,674	76,989	53,638	40,339	125,475	24,774	-	-	-	-	-
Alternative Minimum Income Tax	20%	**$000s**	**196,429**	**18,262**	**15,303**	**10,376**	**4,668**	**938**	**13,015**	**21,135**	**15,398**	**10,728**	**8,068**	**25,095**	**4,955**	**-**	**-**	**-**	**-**	**-**
Federal CIT Reconciliation																				
Regular Federal Income Tax After NOL		$000s	119,547	13,479	9,245	1,873	-	-	-	8,748	12,718	6,411	2,916	27,606	-	-	-	-	-	-
AMT After NOL	0	$000s	82,834	4,782	6,058	8,503	4,668	938	13,015	12,387	2,680	4,317	5,152	(2,511)	4,955	-	-	-	-	-
Adjusted Federal Income Tax		**$000s**	**202,381**	**18,262**	**15,303**	**10,376**	**4,668**	**938**	**13,015**	**21,135**	**15,398**	**10,728**	**8,068**	**25,095**	**4,955**	**-**	**-**	**-**	**-**	**-**
State Income Tax																				
South Carolina Income Tax	5%	$000s	**17,078**	**1,926**	**1,321**	**268**	**-**	**-**	**-**	**1,250**	**1,817**	**916**	**417**	**3,944**	**-**	**-**	**-**	**-**	**-**	**-**
Total Corporate Income Tax		**$000s**	**219,459**	**20,187**	**16,623**	**10,643**	**4,668**	**938**	**13,015**	**22,385**	**17,215**	**11,643**	**8,484**	**29,039**	**4,955**	**-**	**-**	**-**	**-**	**-**
Loss Carry Forward																				
Opening Balance	0	$000s	-	-	-	-	-	(19,412)	(52,541)	(33,731)	-	-	-	-	-	(1,417)	(11,417)	(19,417)	(19,917)	(20,217)
Additions		$000s	(59,627)	-	-	-	(19,412)	(33,130)	-	-	-	-	-	-	(1,417)	(10,000)	(8,000)	(500)	(300)	-
Losses Used		$000s	59,627	-	-	-	-	-	18,811	33,731	-	-	-	-	-	-	-	-	-	-
Closing Balance		$000s	(20,217)	-	-	-	(19,412)	(52,541)	(33,731)	-	-	-	-	-	(1,417)	(11,417)	(19,417)	(19,917)	(20,217)	(20,217)
Depletion																				
1) Percentage Depletion																				
Gross Income From Mining Method																				
Gross Revenue		$000s	3,709,892	305,090	289,511	268,865	247,052	218,552	267,355	271,302	234,909	204,107	184,964	269,296	151,344	-	-	-	-	-
Less Royalty		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Adj. Gross Income		**$000s**	**3,709,892**	305,090	289,511	268,865	247,052	218,552	267,355	271,302	234,909	204,107	184,964	269,296	151,344	-	-	-	-	-
GIFM Depletion	15%	**$000s**	**399,791**	**45,764**	**43,427**	**40,330**	**37,058**	**32,783**	**40,103**	**40,695**	**35,236**	**30,616**	**27,745**	**40,394**	**22,702**	**-**	**-**	**-**	**-**	**-**
Net Income From Mining Method																				
Gross Income		$000s	3,709,892	305,090	289,511	268,865	247,052	218,552	267,355	271,302	234,909	204,107	184,964	269,296	151,344	-	-	-	-	-
Less Operating Costs		$000s	(1,812,948)	(139,848)	(134,202)	(139,931)	(145,041)	(135,109)	(119,587)	(121,138)	(119,655)	(114,520)	(109,605)	(91,931)	(85,699)	(10,000)	(8,000)	(500)	(300)	-
Less Depreciation		$000s	(967,965)	(75,660)	(81,426)	(81,411)	(83,213)	(82,806)	(85,791)	(43,643)	(38,957)	(37,904)	(37,647)	(51,732)	(43,718)	-	-	-	-	-
Taxable Income Before Depletion		**$000s**	**928,979**	89,583	73,883	47,523	18,798	636	61,977	106,522	76,297	51,683	37,712	125,633	21,927	(10,000)	(8,000)	(500)	(300)	-
NIFM Limit	50%	**$000s**	**317,263**	**44,792**	**36,941**	**23,761**	**9,399**	**318**	**30,989**	**53,261**	**38,149**	**25,842**	**18,856**	**62,817**	**10,963**	**-**	**-**	**-**	**-**	**-**
Smaller Allowed Percent Depletion		**$000s**	**405,354**	**44,792**	**36,941**	**23,761**	**9,399**	**318**	**30,989**	**40,695**	**35,236**	**25,842**	**18,856**	**40,394**	**10,963**	**-**	**-**	**-**	**-**	**-**
2) Cost Depletion																				
Acquisition Cost Basis	642,000	$000s	-	503,984	451,188	401,088	354,561	311,808	273,987	227,721	180,772	140,121	104,800	72,792	26,190	0	0	0	0	0
LoM Recovered Ounces	2,854	koz	2,854	235	223	207	190	168	206	209	181	157	142	207	116	-	-	-	-	-
Cost Depletion Allowed		**$000s**	**642,000**	**52,796**	**50,100**	**46,527**	**42,753**	**37,821**	**46,266**	**46,949**	**40,651**	**35,321**	**32,008**	**46,602**	**26,190**	**-**	**-**	**-**	**-**	**-**
Larger of Allowed Depletion		**$000s**	**642,000**	**52,796**	**50,100**	**46,527**	**42,753**	**37,821**	**46,266**	**46,949**	**40,651**	**35,321**	**32,008**	**46,602**	**26,190**	**-**	**-**	**-**	**-**	**-**
Domestic Productions Activities Deduction																				
EBITDA																				
Net Free Cash Flow before Tax		$000s	1,533,576	157,421	134,967	118,271	74,812	67,903	114,051	144,217	115,921	75,750	62,769	176,979	70,128	(8,667)	(8,000)	(500)	(300)	-
Capital Cost Addition		$000s	560,580	13,297	25,820	15,859	32,047	19,977	37,880	10,849	4,172	18,058	16,329	4,154	-	-	-	-	-	-
Depreciation Deduction		$000s	(967,965)	(75,660)	(81,426)	(81,411)	(83,213)	(82,806)	(85,791)	(43,643)	(38,957)	(37,904)	(37,647)	(51,732)	(43,718)	-	-	-	-	-
Depletion Deduction		**$000s**	(642,000)	(52,796)	(50,100)	(46,527)	(42,753)	(37,821)	(46,266)	(46,949)	(40,651)	(35,321)	(32,008)	(46,602)	(26,190)	-	-	-	-	-
Net Taxable Income / (NOL)		**$000s**	**484,191**	**42,263**	**29,260**	**6,192**	**(19,107)**	**(32,747)**	**19,874**	**64,474**	**40,486**	**20,584**	**9,443**	**82,799**	**220**	**(8,667)**	**(8,000)**	**(500)**	**(300)**	**-**
Net Taxable Income Amount	9%	**$000s**	50,644	3,804	2,633	557	-	-	1,789	5,803	3,644	1,853	850	7,452	20	-	-	-	-	-
Mining Salaries and Wages Amount	50%	**$000s**	153,200	15,386	14,139	14,870	16,281	12,553	7,842	8,973	9,209	8,446	7,713	5,079	4,529	-	-	-	-	-
DPA Deduction	**YES**	**$000s**	**33,913**	**3,804**	**2,633**	**557**	**-**	**-**	**1,789**	**5,803**	**3,644**	**1,853**	**850**	**5,079**	**20**	**-**	**-**	**-**	**-**	**-**
Depreciation																				
1) Straight Line																				
a) Undepreciated Initial Construction																				
Undepreciated Balance as of Jan 2017	417,000	$000s																		
Depreciation Term	10	Years																		
Initial Construction Depreciation		**$000s**	**417,000**	**41,700**	**41,700**	**41,700**	**41,700**	**41,700**	**41,700**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
b) Mobile Equipment - Project																				
Total OP+UG Mobile Equipment Capex		$000s	150,770	4,739	1,952	3,459	5,053	17,965	4,200	9,957	-	9,300	11,465	2,500	-	-	-	-	-	-
Calendar Year		-		2049	2050	2051	2052	2053	2054	2055	2056	2057	2058	2059	2060	2061	2062	2063	2064	2065
Total Capex by Year		$000s	150,770	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Depreciation Term	7	Years																		
Total Annual Depreciation By Year	0	$000s	150,770	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)
Total MC Depreciation by Period		$000s	150,770	12,131	12,410	12,904	13,626	16,193	7,927	6,761	6,084	7,133	8,277	7,912	5,346	4,746	3,324	3,324	1,995	357

YEAR				2017	2017	2017	2017	2017	2017	2017	2017	2017	2017	2017	2017
Quarter				1Q2017	1Q2017	1Q2017	2Q2017	2Q2017	2Q2017	3Q2017	3Q2017	3Q2017	4Q2017	4Q2017	4Q2017
Month				Jan-17	Feb-17	Mar-17	Apr-17	May-17	Jun-17	Jul-17	Aug-17	Sep-17	Oct-17	Nov-17	Dec-17
Days Per Period				31	28	31	30	31	30	31	31	30	31	30	31
Year Counter			LoM Total	1	1	1	1	1	1	1	1	1	1	1	1
Project Timeline	EOP @ 5%	US$ & Metric Units	or Avg.	1	2	3	4	5	6	7	8	9	10	11	12
2) Units of Production Method - Project															
Expansion		$000s	314,495	6,516	6,035	6,406	8,634	6,574	6,548	7,946	6,429	6,636	-	-	574
Sustaining		$000s	246,085	6,030	992	(16)	3,997	4,420	4,288	3,110	2,454	2,824	1,515	321	20
Plant Capex Timing Adj.		$000s	-												
Less LoM OP+UG Mobile Equipment Capex		$000s	(150,770)	-	-	-	-	-	-	-	-	-	-	-	-
Less LoM Land Purchases		$000s	(9,615)	-	-	-	-	-	-	-	(10)	(565)	-	-	-
Total Capital		$000s	400,195	12,546	7,028	6,390	12,631	10,994	10,836	11,055	8,872	8,895	1,515	321	594
LoM Recovered Oz Produced		**koz**	**2,854**	**4.0**	**10.2**	**10.9**	**12.0**	**12.9**	**12.4**	**12.2**	**11.7**	**11.0**	**10.8**	**10.4**	**10.7**
Year	**Capital/yr**	**Gold Prod/yr**	**Total Dep.**												
2017	91,679	129	91,679	128.1	326.8	351.2	384.4	413.2	399.7	393.1	377.1	351.8	347.1	335.1	344.8
2018	66,330	143	66,330												
2019	16,800	163	16,800												
2020	43,690	178	43,690												
2021	26,807	235	26,807												
2022	59,460	223	59,460												
2023	12,401	207	12,401												
2024	26,994	190	26,994												
2025	2,012	168	2,012												
2026	33,680	206	33,680												
2027	892	209	892												
2028	4,172	181	4,172												
2029	8,758	157	8,758												
2030	4,864	142	4,864												
2031	1,654	207	1,654												
2032	-	116	-												
2033	-	-	-												
2034	-	-	-												
2035	-	-	-												
2036	-	-	-												
Total Calculated Depreciation	400,195	2,854	400,195	128.1	326.8	351.2	384.4	413.2	399.7	393.1	377.1	351.8	347.1	335.1	344.8
Total Project Calculated Depreciation			**550,965**	**128**	**327**	**351**	**384**	**413**	**400**	**393**	**377**	**352**	**347**	**335**	**345**
Total Project Calc'd Dep w/ EOM Writeoff			**550,965**	**128**	**327**	**351**	**384**	**413**	**400**	**393**	**377**	**352**	**347**	**335**	**345**

YEAR	2018	2018	2018	2018	2018	2018	2018	2018	2018	2018	2018	2018
Quarter	1Q2018	1Q2018	1Q2018	2Q2018	2Q2018	2Q2018	3Q2018	3Q2018	3Q2018	4Q2018	4Q2018	4Q2018
Month	Jan-18	Feb-18	Mar-18	Apr-18	May-18	Jun-18	Jul-18	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18
Days Per Period	31	28	31	30	31	30	31	31	30	31	30	31
Year Counter	2	2	2	2	2	2	2	2	2	2	2	2
Project Timeline	13	14	15	16	17	18	19	20	21	22	23	24
Expansion	-	-	17,483	-	-	13,730	-	-	13,732	2	2	13,732
Sustaining	5,169	1,959	962	677	609	879	654	162	292	62	162	62
Less LoM OP+UG Mobile Equipment Capex	-	-	-	-	-	-	-	-	-	-	-	-
Less LoM Land Purchases	(4,000)	-	-	-	-	-	-	-	-	-	-	-
Total Capital	1,169	1,959	18,445	677	609	14,609	654	162	14,024	64	164	13,794
LoM Recovered Oz Produced	**14.7**	**12.5**	**13.2**	**12.1**	**12.2**	**11.5**	**11.7**	**11.8**	**11.1**	**11.1**	**10.4**	**10.8**
2017	470.7	401.5	424.8	388.9	391.3	368.5	374.6	380.7	358.2	355.2	333.7	346.4
2018	356.7	304.3	322.0	294.7	296.6	279.3	283.8	288.5	271.4	269.2	252.9	262.5
Total Calculated Depreciation	827.4	705.8	746.8	683.6	687.9	647.7	658.4	669.1	629.6	624.4	586.6	608.9
Total Project Calculated Depreciation	**827**	**706**	**747**	**684**	**688**	**648**	**658**	**669**	**630**	**624**	**587**	**609**
Total Project Calc'd Dep w/ EOM Writeoff	**827**	**706**	**747**	**684**	**688**	**648**	**658**	**669**	**630**	**624**	**587**	**609**

YEAR	2019	2019	2019	2019	2020	2020	2020	2020
Quarter	1Q2019	2Q2019	3Q2019	4Q2019	1Q2020	2Q2020	3Q2020	4Q2020
Month	Mar-19	Jun-19	Sep-19	Dec-19	Mar-20	Jun-20	Sep-20	Dec-20
Days Per Period	90	91	92	92	91	91	92	92
Year Counter	3	3	3	3	4	4	4	4
Project Timeline	25	26	27	28	29	30	31	32
Expansion	56,159	4,394	18,847	15,137	34,363	24,886	18,704	21,028
Sustaining	1,123	544	559	349	1,156	1,182	372	752
Plant Capex Timing Adj.	(1,738)	(3,476)	(4,345)	(8,690)	(13,035)	(17,380)	(8,690)	(1,528)
Less LoM OP+UG Mobile Equipment Capex	(53,210)	-	(8,853)	-	(10,398)	-	(1,317)	(6,403)
Less LoM Land Purchases	-	-	-	-	-	-	-	-
Total Capital	2,334	1,462	6,208	6,796	12,085	8,687	9,069	13,849
LoM Recovered Oz Produced	**39.2**	**40.2**	**40.8**	**42.9**	**41.2**	**40.7**	**41.0**	**55.3**
2017	1,259.2	1,291.4	1,309.5	1,379.7	1,322.8	1,307.4	1,315.9	1,776.2
2018	954.3	978.7	992.4	1,045.6	1,002.5	990.8	997.2	1,346.0
2019	255.1	261.6	265.3	279.5	268.0	264.9	266.6	359.8
2020					743.9	735.2	740.0	998.8
Total Calculated Depreciation	2,468.6	2,531.8	2,567.2	2,704.7	3,337.1	3,298.3	3,319.7	4,480.8
Total Project Calculated Depreciation	**4,685**	**4,748**	**4,784**	**4,921**	**6,201**	**6,162**	**6,183**	**7,344**
Total Project Calc'd Dep w/ EOM Writeoff	**4,685**	**4,748**	**4,784**	**4,921**	**6,201**	**6,162**	**6,183**	**7,344**

YEAR				2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Quarter				4Q2021	4Q2022	4Q2023	4Q2024	4Q2025	4Q2026	4Q2027	4Q2028	4Q2029	4Q2030	4Q2031	4Q2032	4Q2033	4Q2034	4Q2035	4Q2036	4Q2037
Month				Dec-21	Dec-22	Dec-23	Dec-24	Dec-25	Dec-26	Dec-27	Dec-28	Dec-29	Dec-30	Dec-31	Dec-32	Dec-33	Dec-34	Dec-35	Dec-36	Dec-37
Days Per Period				365	365	365	366	365	365	365	366	365	365	365	366	365	365	365	366	365
Year Counter			LoM Total	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
Project Timeline	EOP @ 5%	US$ & Metric Units	or Avg.	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47	48	49
2) Units of Production Method - Project																				
Expansion		$000s	314,495	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining		$000s	246,085	13,297	25,820	15,859	32,047	19,977	37,880	10,849	4,172	18,058	16,329	4,154	-	-	-	-	-	-
Plant Capex Timing Adj.		$000s	-	18,249	40,632															
Less LoM OP+UG Mobile Equipment Capex		$000s	(150,770)	(4,739)	(1,952)	(3,459)	(5,053)	(17,965)	(4,200)	(9,957)	-	(9,300)	(11,465)	(2,500)	-	-	-	-	-	-
Less LoM Land Purchases		$000s	(9,615)	-	(5,040)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital		$000s	400,195	26,807	59,460	12,401	26,994	2,012	33,680	892	4,172	8,758	4,864	1,654	-	-	-	-	-	-
LoM Recovered Oz Produced		**koz**	**2,854**	**234.7**	**222.7**	**206.8**	**190.0**	**168.1**	**205.7**	**208.7**	**180.7**	**157.0**	**142.3**	**207.2**	**116.4**	**-**	**-**	**-**	**-**	**-**
Year	**Capital/yr**	**Gold Prod/yr**	**Total Dep.**																	
2017	91,679	129	91,679	7,539.4	7,154.4	6,644.2	6,105.2	5,400.9	6,606.9	6,704.4	5,805.1	5,043.9	4,570.9	6,654.9	3,740.0	-	-	-	-	-
2018	66,330	143	66,330	5,713.6	5,421.8	5,035.2	4,626.7	4,092.9	5,006.9	5,080.8	4,399.3	3,822.4	3,463.9	5,043.2	2,834.3	-	-	-	-	-
2019	16,800	163	16,800	1,527.3	1,449.3	1,346.0	1,236.8	1,094.1	1,338.4	1,358.2	1,176.0	1,021.8	925.9	1,348.1	757.6	-	-	-	-	-
2020	43,690	178	43,690	4,239.8	4,023.3	3,736.3	3,433.2	3,037.2	3,715.4	3,770.2	3,264.5	2,836.4	2,570.4	3,742.3	2,103.2	-	-	-	-	-
2021	26,807	235	26,807	2,808.3	2,664.9	2,474.8	2,274.1	2,011.7	2,460.9	2,497.3	2,162.3	1,878.8	1,702.5	2,478.8	1,393.1	-	-	-	-	-
2022	59,460	223	59,460		6,602.5	6,131.6	5,634.2	4,984.2	6,097.2	6,187.2	5,357.2	4,654.8	4,218.2	6,141.4	3,451.5	-	-	-	-	-
2023	12,401	207	12,401			1,438.5	1,321.8	1,169.3	1,430.4	1,451.6	1,256.8	1,092.0	989.6	1,440.8	809.7	-	-	-	-	-
2024	26,994	190	26,994				3,254.9	2,879.4	3,522.4	3,574.4	3,094.9	2,689.1	2,436.9	3,547.9	1,993.9	-	-	-	-	-
2025	2,012	168	2,012					244.0	298.5	302.9	262.3	227.9	206.5	300.7	169.0	-	-	-	-	-
2026	33,680	206	33,680						5,687.3	5,771.3	4,997.1	4,341.9	3,934.7	5,728.6	3,219.5	-	-	-	-	-
2027	892	209	892							183.9	159.2	138.4	125.4	182.5	102.6	-	-	-	-	-
2028	4,172	181	4,172								938.2	815.2	738.7	1,075.5	604.4	-	-	-	-	-
2029	8,758	157	8,758									2,207.8	2,000.7	2,912.9	1,637.1	-	-	-	-	-
2030	4,864	142	4,864										1,485.6	2,162.9	1,215.5	-	-	-	-	-
2031	1,654	207	1,654											1,059.1	595.2	-	-	-	-	-
2032	-	116	-												-	-	-	-	-	-
2033	-	-	-																	
2034	-	-	-																	
2035	-	-	-																	
2036	-	-	-																	
Total Calculated Depreciation	400,195	2,854	400,195	21,828.4	27,316.2	26,806.7	27,886.8	24,913.7	36,164.4	36,882.2	32,872.8	30,770.2	29,370.0	43,819.9	24,626.8	-	-	-	-	-
Total Project Calculated Depreciation			**550,965**	**33,960**	**39,726**	**39,711**	**41,513**	**41,106**	**44,091**	**43,643**	**38,957**	**37,904**	**37,647**	**51,732**	**29,973**	**4,746**	**3,324**	**3,324**	**1,995**	**357**
Total Project Calc'd Dep w/ EOM Writeoff			**550,965**	**33,960**	**39,726**	**39,711**	**41,513**	**41,106**	**44,091**	**43,643**	**38,957**	**37,904**	**37,647**	**51,732**	**43,718**	**-**	**-**	**-**	**-**	**-**